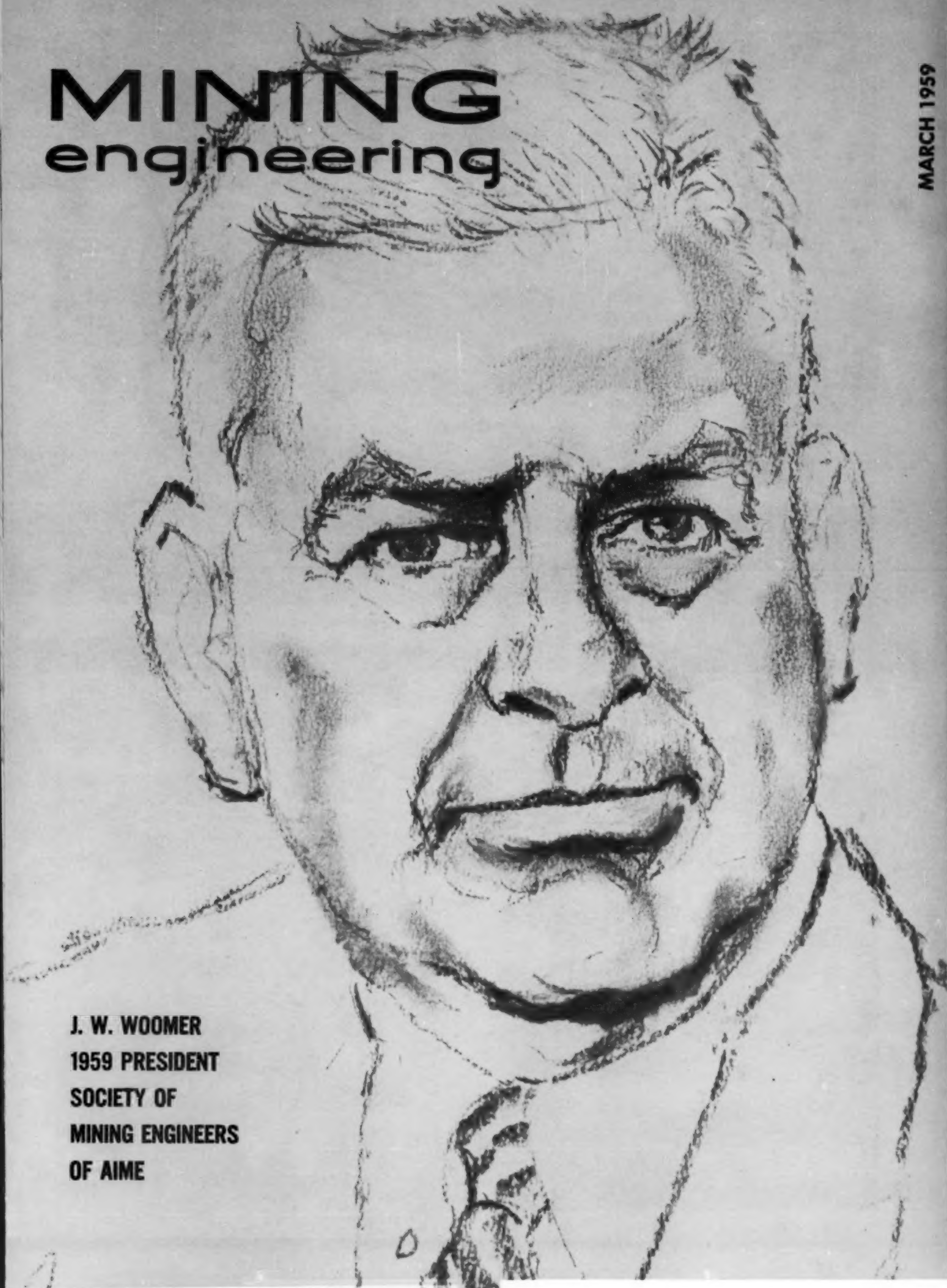
MINING engineering

MARCH 1959



J. W. WOOMER
1959 PRESIDENT
SOCIETY OF
MINING ENGINEERS
OF AIME

COMING EVENTS

Mar. 16-20, 11th Western Metal Congress, American Society for Metals, Los Angeles.

Apr. 3-5, AIME Southwest Alaska, Alaska Sections, annual meeting, Westward Hotel, Anchorage, Alaska.

Apr. 3 or 10, AIME Lehigh Valley Section, technical meeting.

Apr. 5-10, EJC 1959 Nuclear Congress, Public Auditorium, Cleveland.

Apr. 13-15, CIM, annual meeting, Queen Elizabeth Hotel, Montreal.

Apr. 16-18, AIME Pacific Northwest Regional Conference, Olympic Hotel, Seattle.

Apr. 18, AIME Colorado MBD Subsection, Broadmoor Hotel, Colorado Springs, Colo.

Apr. 20-22, Third Rock Mechanics Symposium, tri-sponsors: Colorado School of Mines, Pennsylvania State University, and University of Minnesota; Colorado School of Mines, Golden, Colo.

Apr. 22-23, Lead Industries Assn., 31st annual meeting, Drake Hotel, Chicago.

Apr. 23-24, American Zinc Inst., 41st annual meeting, Drake Hotel, Chicago.

May 8-10, Fourth Annual Uranium Symposium, AIME Uranium Section, Moab, Utah.

May 11-14, American Mining Congress, Coal Show, Cleveland.

May 21-22, Lake Superior Mines Safety Council, 35th annual conference, Hotel Duluth, Duluth.

June, AIME Lehigh Valley Section, field trip, New Jersey Zinc Co., Friedensville, Pa.

June 28-July 1, Rocky Mountain Coal Mining Inst., annual meeting, Antlers Hotel, Colorado Springs, Colo.

Sep. 14-17, American Mining Congress, Metal Mining & Industrial Minerals Convention, Denver.

Sept. 24-26, SME Industrial Minerals and Coal Divisions, joint meeting, Bedford Springs, Pa.

October, AIME Lehigh Valley Section, technical meeting.

Oct. 8-10, Exploration Drilling Symposium, tri-sponsors: Colorado School of Mines, Pennsylvania State University, and University of Minnesota; Pennsylvania State University, University Park, Pa.

Oct. 27-29, 1959 AIME-ASME Joint Solid Fuels Conference, Netherland-Hilton Hotel, Cincinnati.

Nov. 9-12, Society of Exploration Geophysicists, annual meeting, Biltmore Hotel, Los Angeles.

Dec. 4, AIME Lehigh Valley Section, ladies' night.

Dec. 7, AIME Arizona Section, annual meeting, Tucson, Ariz.

MINING engineering

Vol. 11 NO. 3 **MARCH 1959**

COVER *Artist Herb McClure salutes the 1959 Society of Mining Engineers' President—J. W. Woomer—on this month's cover. For a profile of "Jerry," world renowned consulting mining engineer from Pittsburgh, turn to page 287.*

ANACONDA'S BERKELEY OPEN PIT

TECHNICAL ARTICLES

DEPARTMENTS

FEATURED ITEMS

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Go Ford-ward for greater payload...power

"Our Ford trucks haul up to a ton-and-a-half more payload per trip"

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"We keep our Ford T-700's in top condition year round, and it pays off. We get a higher resale price when we trade every two years. Fords have the ability to perform under the rugged conditions in our work. Power steering on our tandem dumps makes them easy to handle on-or off-the road.

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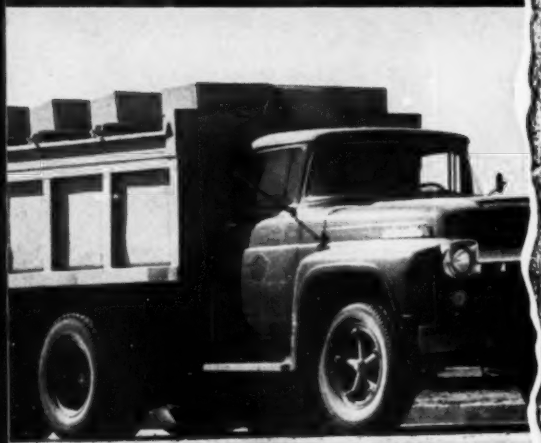

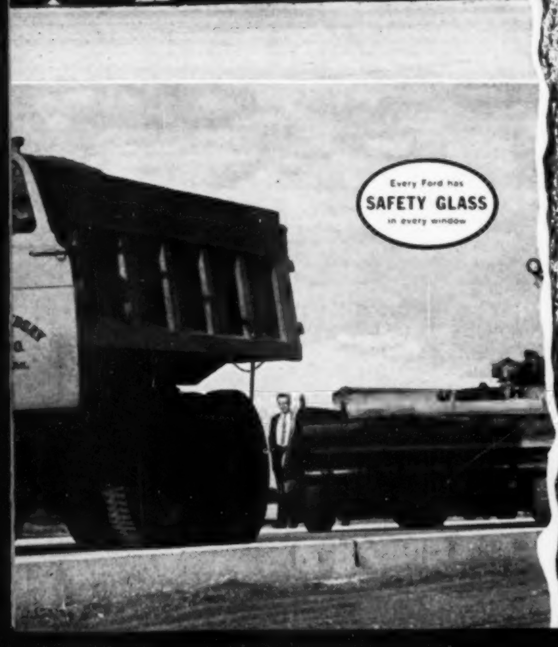

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Detroit 31, Michigan

'59 Ford Pickups Win Economy Showdown U.S.A.

—average 25.2% better gas mileage!

Impartial tests of the 1959 pickup models of all six makes prove conclusively that Ford's ½-ton pickups equipped with Short Stroke Sixes are the economy champs for '59.

HOW TESTS WERE MADE

Standard six-cylinder models of the six leading half-ton pickups first were put through exhaustive road trials. All '59 trucks—Ford and competitive—were bought from dealers, just as you would buy them. After at least 600 miles break-in, all were brought up to manufacturer's recommended specifications.

The trucks were then tested — by America's leading independent automotive testing firm—at constant speeds of 30, 45 and 60 miles an hour. Next came stop-and-go tests, ranging from moderate city traffic to normal retail delivery operation. Acceleration rates were carefully timed in each gear to insure accurate results for all makes.

HOW NEW '59 SIXES RATE IN GAS MILEAGE						
'59 FORD SIXES GIVE	25.2% more miles per gallon than Make "C"	31.1% more miles per gallon than Make "I"	9.6% more miles per gallon than Make "G"	42.6% more miles per gallon than Make "D"	22.0% more miles per gallon than Make "S"	25.2% more miles per gallon than the average of all makes

The '59 Ford Sixes, *in every test*, averaged more miles per gallon than every other make! Combining all tests, the '59 Fords led the average of all other '59 pickups by 25.2%.

WHAT'S THE SECRET?

How can a '59 Ford Six make four gallons do the work of five in other trucks?

First, of all pickup Sixes, only Ford has modern Short Stroke design. This new type of engine is basically far more efficient than long-stroke Sixes of other pickups. Example: Ford's Six delivers more usable horsepower than any other pickup Six.

Second, to this modern engine Ford has added a new economy carburetor. By metering fuel more precisely in both low- and high-speed ranges, Ford's new carburetor boosts gasoline mileage in every type of driving. And Ford's *Economy Carburetor is standard at no extra cost.*

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PERSONNEL

THESE items are listings of the Engineering Societies Personnel Service Inc. This Service, which cooperates with the national societies of Civil; Electrical; Mechanical; Mining, Metallurgical, and Petroleum Engineers, is available to all engineers, members and nonmembers, and is operated on a nonprofit basis. If you are interested in any of these listings, and are not registered, you may apply by letter or resume and mail to the office nearest your place of residence, with the understanding that should you secure a position as a result of these listings you will pay the regular employment fee of 5 pct of the first year's salary if a nonmember, or 4 pct if a member. Also, that you will agree to sign our placement fee agreement which will be mailed to you immediately, by our office, after receiving your application. In sending applications be sure to list the key and job number. When making application for a position, include 8¢ in stamps for forwarding application to the employer and for returning when possible. A weekly bulletin of engineering positions open is available at a subscription rate of $3.50 per quarter or $12 per annum for members, $4.50 per quarter or $14 per annum for nonmembers, payable in advance. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18**; 57 Post St., **San Fencisco**; 84 E. Randolph St., **Chicago 1**.

MEN AVAILABLE

Mining Engineer, B.Sc. in mining engineering, age 29. Has experience in open pit and underground mining, research engineering, structural engineering, and cost estimating. Desires responsible position, preferably in Canada or northern U. S. M-462.

Mine Superintendent, B.S. in mining, age 34. Gained eight years experience in varied operational, administrative positions in the U. S. and Latin America. Location desired, Mexico. M-582-San Francisco.

Mine Superintendent, age 54. Licensed Professional Engineer and land surveyor. Twenty-five years experience underground and open pit mining, exploration, development, and operation. Supervisory or engineering position desired, U. S. or foreign. Available with two weeks notice. M-695-San Francisco.

Manager or Superintendent, age 44. Has 20 years of experience in operation, examination, and development work at small to medium sized base and precious metal mines. Also some metallurgical, plant design, operation, and allied mechanical and construction work. Prefers the U. S. M-463.

Geologist, B.S., age 31. Worked for seven years in all phases of mineral exploration, principally in uranium. Has ability to write. Gained some supervisory experience in addition to research. Accustomed to dealing with the public. Location, immaterial. M-728-San Francisco.

Mining Engineer, B.S., age 42. For 15 years has been gaining sound and responsible experience in all phases of exploration, including geologic mapping, planning, and supervising exploration and development, estimating reserves, economics, negotiating agreements, etc. Broad range of experience and perspective. Excellent references. Prefers western or southern U. S. M-370-San Francisco.

Construction Engineer, B.S. in mining, age 27. Has four and one half years work in highway and light construction. Location desired, foreign. M-464.

Metallurgist, Mining Engineer, mining engineering degree, age 35, married, no children. Three years working underground as chief engineer, two years as assistant to chief metallurgist in charge of the projection of a 4000 tpd mill for the treatment of columbium ores in West Africa. Location, immaterial. M-465.

Vice President or General Manager, B.S. in mining engineering, B.S. in electrical engineering, age 45. Over 20 years spent in heavy industry. Experienced in all phases of management, especially engineering, sales, and operations. Location, immaterial. M-466.

Manager or Superintendent, B.S. in mining and metallurgy, age 40. Has experience in mill, mill machinery design, construction and operation; exploration, and exploitation of mine and mine plants. Has 17 years experience, eight spent in Latin America, with working Mexican passport. Prefers Mexico, southwest, or South America. M-657-San Francisco.

POSITIONS OPEN

Recent Graduate Mining Engineer, with degree in mining engineering or ore dressing, for a combination field and laboratory trouble shooting assignment in the wet processing phosphate rock operation. Location, South. W6655.

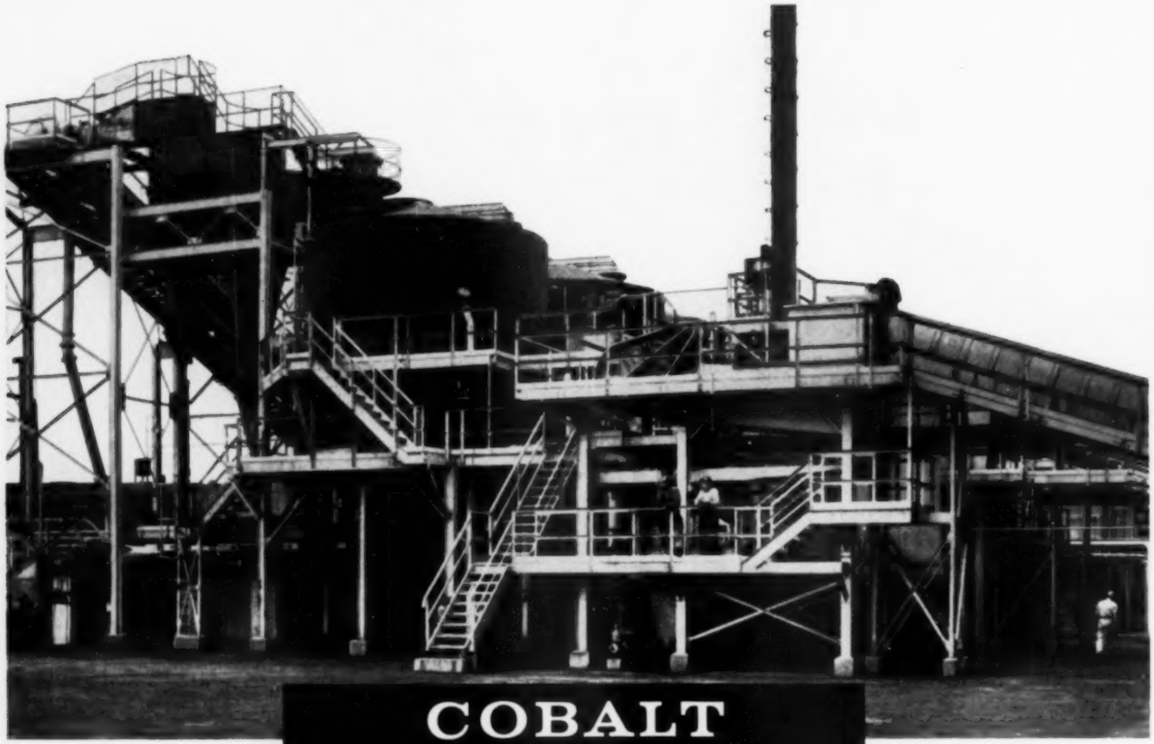
Mine Superintendent with production and management experience in mining or uranium ore, to take charge of 1000 tpd operation at 50-100,000 t proven property. Salary, $9000 a year plus living quarters. Location, Arizona. W6504.

District Sales Representative for manufacturer of heavy construction and mining equipment. Must be experienced in application of excavating and loading equipment. Sales experience desirable. Headquarters, New York. W6179.

Mining Engineers, two recent graduates, one with possibly one or two years experience, to do surveying and mapping, etc. One applicant who has a chemical background would be useful. Salaries open, depending upon experience. Location, upstate New York. W6972.

Maintenance Engineer, graduate mining or mechanical engineer, for phosphate rock installation. Will consider men with coal mining or other underground mining experience in maintenance. Location, South. W6971.

Assistant to President, mining, chemical engineering, or business administration graduate, with at least seven years managerial or staff experience in metal mine production. Salary, $25,000- to $40,000 a year. Location, New York. W6787.



COBALT FROM PYRITES

Versatile Dorrco FluoSolids System adds another "plus" at East Coast mill

Installed in 1952 primarily for production of SO_2 gas for acid making and a calcine for iron manufacture, the Dorrco FluoSolids system at a well known East Coast steel mill gives proof of the versatility of the fluidizing technique.

Pyrite received by the mill from one source of supply has a valuable cobalt content. A single roasting operation not only promotes efficient preferential cobalt sulfatization that results in an average 90 percent cobalt extraction at the leaching plant, but also produces SO_2 in sufficient strength for sulfuric acid manufacture as well as calcine for blast furnace charging. The success of this operation, never before attempted in commercial practice, is an outstanding example of the many processing opportunities offered by the Dorrco FluoSolids system.

The installation at this plant consists of three 18′ diam. reactors with pulping and holding tanks, cyclones and other auxiliary equipment. Currently one reactor is used to handle the cobalt-bearing concentrate.

Applications of the Dorrco FluoSolids system in other industries include arsenopyrite gold roasting, zinc concentrate roasting, providing a sulfating roast for copper-zinc concentrates, roasting sulfides for making cooking liquor in sulfite paper mills and limestone calcination.

If you'd like more information on this significant advance in roasting techniques, write to Dorr-Oliver Incorporated, Stamford, Conn.

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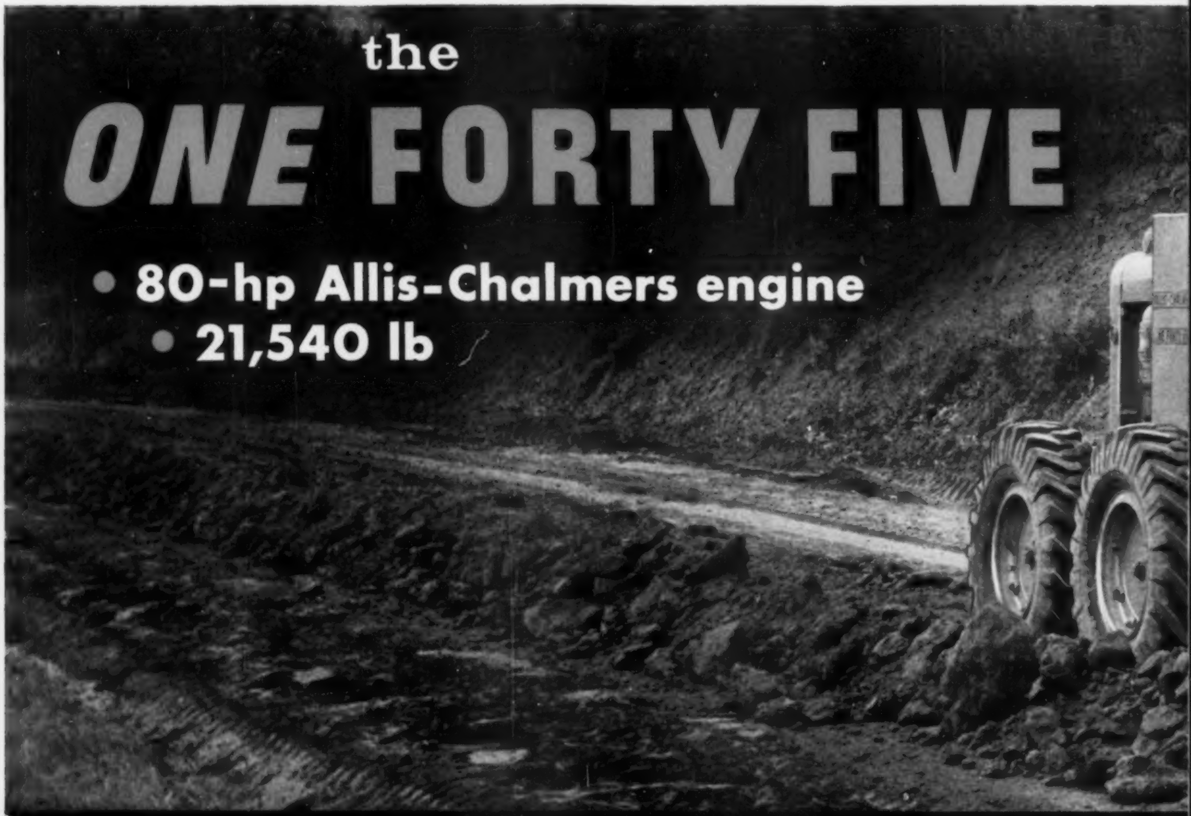
Allis-Chalmers announces a new motor grader in the 80-hp class



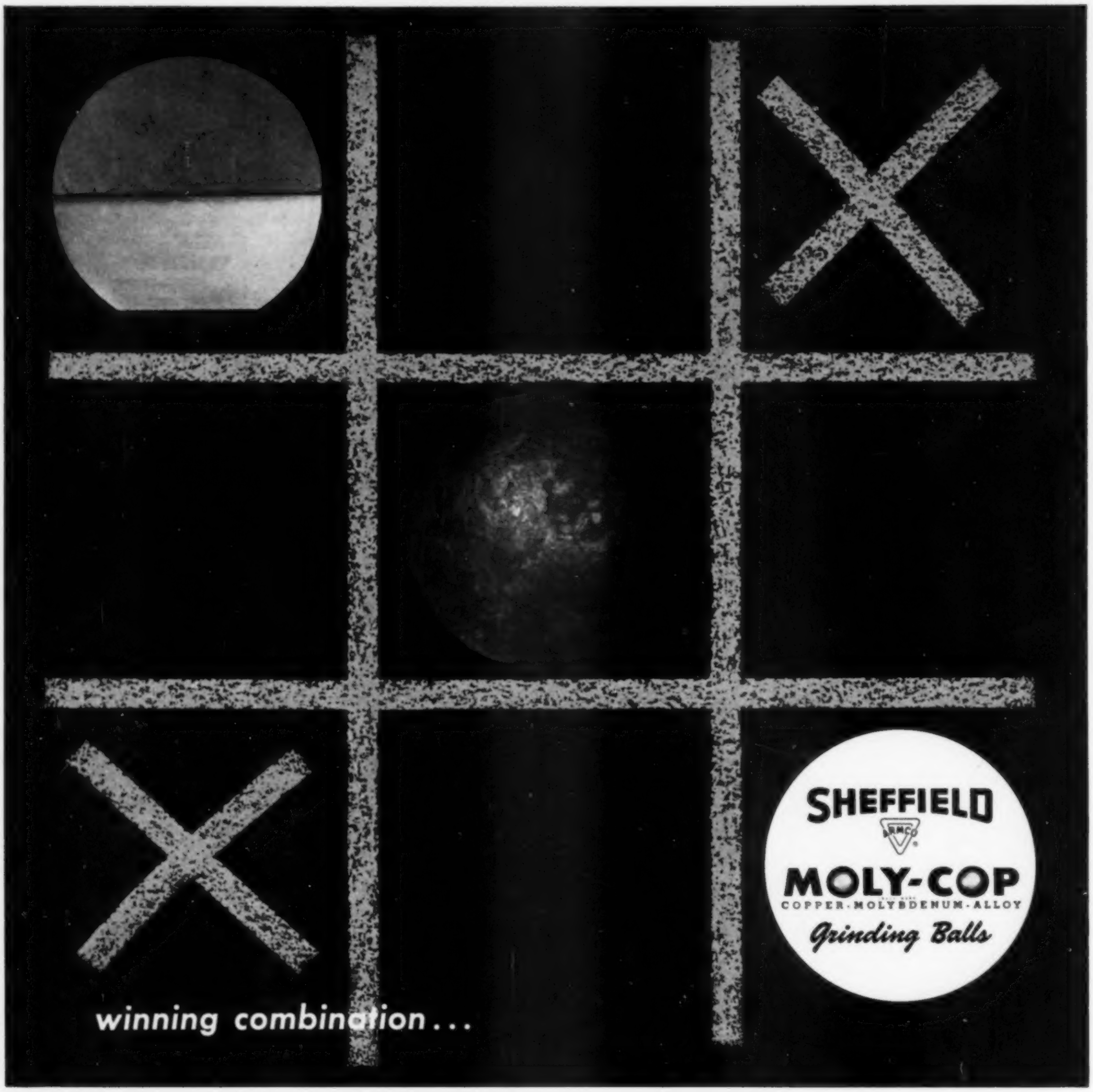
... Another outstanding motor grader joins the Allis-Chalmers line the ONE FORTY FIVE

MODEL D
58-hp Allis-Chalmers engine
8,800 to 11,450 lb

***ONE* FORTY FIVE**
80-hp Allis-Chalmers engine
21,540 lb

FORTY FIVE
120-hp Allis-Chalmers engine
23,800 lb

move ahead with **ALLIS-CHALMERS**..



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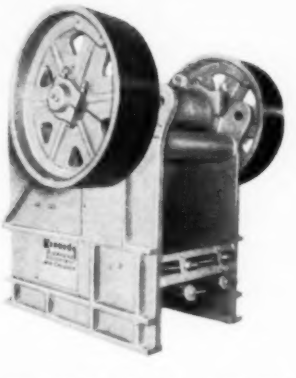
KENNEDY JAW CRUSHERS



The KENNEDY Overhead Eccentric Jaw Crusher features a massive, one-piece, arc welded steel plate frame, and is available in standard sizes from 10" x 36" to 36" x 42".

Long life—dependable service—low maintenance—minimum power—high production—make KENNEDY Jaw Crushers the first choice of all who buy because of proven performance and low year-to-year cost.

Exclusive features of the KENNEDY Jaw Crusher

- Cast Meehanite or steel frame.
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An International Directory of Engineering Source Material

BOOKS

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Technology of Columbium, edited by B. W. Gonser and E. M. Sherwood, *John Wiley and Sons Inc.*, 120 pp., $7, 1958—A summary of the present status of knowledge concerning columbium. Economic and supply aspects, properties, analytical problems are covered in 17 papers presented at a symposium of the Electrochemical Soc.

Company Investigations of Automatic Data Processing, by Peter B. Laubach, *Div., of Research*, Harvard Business School, Soldiers Field, Boston 63, Mass., $3, 1957—Reports of a series of studies to continue indefinitely covering data processing systems used by business organizations. The approaches taken by various companies examining the need for automatic equipment are discussed.

Basic Geology for Science and Engineering, by Edward C. Dapples, *John Wiley & Sons Inc.*, 620 pp., tables and illustrations, $9.50, 1959—A systematic analysis of processes basic to an understanding of physical geology, this text is written by a professor of geology at Northwestern University. Each chapter includes observation and grouping of data, classification of data into forms of graphs and tables, and interpretation of the relationships observed in order to formulate significant principles of geologic behavior.

A Dictionary of Metallurgy, by A. D. Merriman, *Pitman Publishing Corp.*, 2 W. 45th St., New York 36, N. Y., 416 pp., 283 illustrations, 236 tables, $25, 1959—Nearly 7000 definitions make this one of the most comprehensive works of its kind ever to appear in print. Important tables cover common abbreviations and symbols, abbreviations used in foreign literature, and physical properties of metals. The most recently evolved terms of atomic energy and specialized fields of the fabrication of metals are carefully defined.

Fine Particle Measurement, by Clyde Orr, Jr., and J. M. Dallavalle, *The Macmillan Co.*, 370 pp., $10.50, 1959—The specialty of fine particle measurement, as it has grown from the field of particle technology, is considered for its critical influence on manufacturing processes and commercial products such as cement, paint, and innumerable foods. Control of size, surface, and volume according to modern methods now available is discussed.

Rock Pressure in Mines, by E. De St. Q. Isaacson, *Mining Publications Ltd.*, 482 Salisbury House, London E.C. 2, 222 pp., $6.32, 1958—This volume attempts to set forth principles, both theoretical and practical, that govern the behavior of pressure in underground workings. The stresses surrounding an excavation are, too often, unknown quantities, as is the strength of the rock itself and the various support materials used. The progressively increasing depths of mining operations demand further study of pressure and control

The Economics of the Coal Industry, by Hubert E. Risser, *Bureau of Business Research*, School of Business, University of Kansas, 188 pp., 65 tables, 11 figs., 8 appendix figs., $3, 1958—Dr. Risser selected the coal industry as a Ph.D. thesis topic and provided the basic data for this report. He carefully analyzes and explores the industry and the characteristics contributing to its chronic ills.

Minerals Yearbook, 1957, Vol. 1, Metals and Minerals (except fuels), U. S. Government Printing Office, Superintendent of Documents, Washington 25, D. C., 1376 pp., $3.50, 1958—This volume includes chapters on metal and nonmetal mineral commodities, except fuels, with a review, a statistical summary, and chapters on mining, metallurgical technology, employment, and injuries. An additional chapter in the 1957 volume 1 compares Bureau of Mines mineral-commodity production data for 1954 with those presented in the 1954 Census of Mineral Industries reports published by the U. S. Dept. of Commerce.

Manual on Rock Blasting, Supplement No. 6, *Atlas Copco Eastern Inc.*, 610 Industrial Ave., Paramus, N. J., $4.50, 1958—The volume includes the three articles: *Rock Drilling Methods and their Fields of Use* by I. Janelid; *Bench Blasting on Construction Projects in Sweden* by T. Bjarnerull, and *Loading in Swedish Underground Mines* by C. B. Berglund and C. Westlund.

Bibliography of the Liquid-Solid Cyclone 1939 to 1957, by O. F. Tangel, R. J. Brison, and D. A. Jacobs, *Battelle Memorial Inst.*, 505 King Ave., Columbus 1, Ohio, 18 pp., gratis, 1958—This is an enlarged and revised bibliography completed by the minerals beneficiation specialists at Battelle Memorial Inst. It lists 243 articles on the liquid-solid cyclone that were published through 1957, with some early 1958 entries included.

Bibliography and Index of Literature on Uranium and Thorium and Radioactive Occurrences in the U. S. Parts 1-5, *Geological Society of America*, 419 W. 117th St., New York 27, N. Y. **Part 1:** Arizona, Nevada, and New Mexico, 33 pp., 25¢, 1953; **Part 2:** California, Idaho, Montana, Oregon, Washington, and Wyoming, 70 pp., 25¢, 1953; **Part 3:** Colorado and Utah, 124 pp., 50¢, 1954; **Part 4:** Arkansas Iowa, Kansas, Louisiana, Minnesota, Missouri, Nebraska, North Dakota, Oklahoma, South Dakota, and Texas, 70 pp., 50¢, 1955: **Part 5:** Connecticut, Delaware, Illinois, Indiana, Maine, Maryland, Massachusetts, Michigan, New Hampshire, New Jersey, New York, Ohio, Pennsylvania, Rhode Island, Vermont, and Wisconsin, 472 pp., $6.75, 1958. Each part also includes a bibliography, gazatteer, geographical index, and subject index, with detailed geological information for each area covered.

STATE PUBLICATIONS

Alaska

Order From:

School of Mines
P. O. Box 498
College, Alaska

Introductory Prospecting and Mining, by Leo Mark Anthony, $2.50, 1958.

Arizona

Order From:

Arizona Bureau of Mines
University of Arizona
Tucson, Ariz.

Geologic Map of Yavapai County, 75¢, 1958.

California

Order From:

California Div. of Mines
Ferry Bldg.
San Francisco 11, Calif.

Geologic Guidebook of the San Francisco Bay Counties: History, Landscape, Geology, Fossils, Minerals, Industry, and Routes to Travel, Bulletin 154, $2.50, reprint 1959.

Death Valley Sheet, $1.50.

Available Publications, California Division of Mines, November 1958, gratis.

Index to Geologic Maps of California to Dec. 31, 1956, Special Report 52, compiled by R. C. Strand, J. B. Koenig, and C. W. Jennings, $1, 1958.

Plants as a Guide to Mineralization, Special Report 50, by Donald Carlisle and George B. Cleveland, 50¢.

California Journal of Mines and Geology, Vol. 54, No. 4, final issue, $1, 1958.

Report of the State Mineralogist, formerly the **California Journal of Mines and Geology**, published annually rather than quarterly beginning January 1959.

Order From:

Bureau of Mines
Div. of Mineral Industries
Region II, 420 Custom House
555 Battery St.
San Francisco 11, Calif.

Mineral Production in California, Area Report II-29, gratis.

Connecticut

Order From:

Robert C. Sale
State Librarian
State Library
Hartford 15, Conn.

The Bedrock Geology of the Danbury Quadrangle, Quadrangle Report No. 7, by James W. Clarke, $1, 1958.

Petrogenesis of the Voluntown and Oneco Quadrangles, Bulletin No. 89, by Ralph M. Perhac, 50¢, 1958.

The Preston Gabbro and the Associated Metamorphic Gneisses, New London County, Conn., Bulletin No. 88, by Charles B. Sclar, $1.50, 1958.

The Bedrock Geology of the Guilford 15-Minute Quadrangle and a portion of the New Haven Quadrangle, Bulletin No. 86, by Harry M. Mikami and Ralph E. Digman, $1.50, 1957.

Florida

Order From:

Florida Engineering and Industrial
Experiment Station
University of Florida
Gainesville, Fla.

Transactions of the Florida Seminars on Spectroscopy 1955-1957, Bulletin No. 100, Engineering Progress at the University of Florida Vol. XII, No. 11, $4, 1958.

Georgia

Order From:

Georgia State Div. of Conservation
Dept. of Mines, Mining and Geology
Atlanta, Ga.

The Geology of Hart County, Georgia, Bulletin No. 67, by Willard Huntinton Grant, $2, 1958.

Idaho

Order From:

Idaho Bureau of Mines and Geology
Moscow, Idaho

Geology and Mineral Resources of Ada and Canyon Counties, 1958 County Report No. 3, by C. N. Savage, $1.50, 1958.

Gold-Bearing Gravels Near Murray, Idaho, Pamphlet No. 116, by Wakefield Dort, Jr., 75¢, 1958.

Uranium, Thorium, Columbium, and Rare Earth Deposits in the Salmon Region, Lemhi County, Idaho, Pamphlet No. 115, by Alfred L. Anderson, $1.25, 1958.

Petrography, Mineralogy, and Origin of Phosphate Pellets in the Phosphoria Formation, Pamphlet No. 114, by G. Donald Emigh, $1, 1958.

Outline of the Geology of Idaho, Bulletin No. 15, by Clyde P. Ross and J. D. Forrester, $1.50, 1958.

The Mineral Industry of Idaho, Area Report B-61, gratis, available from Bureau of Mines, Div. of Mineral Industries, Region I, PO Box 492, Albany, Ore.

Illinois

Order From:

State Geological Survey
Natural Resources Bldg.
Urbana, Ill.

Some Plastic Properties of Pastes Made from Hydrated Dolomitic and High-Calcium Limes, Circular 261, by D. L. Deadmore and J. S. Machin, gratis, 1958.

Strippable Coal Reserves of Illinois, Part 2—Jackson, Monroe, Perry, Randolph, and St. Clair Counties, Circular 260, by William H. Smith, gratis, 1958.

Three Ostracode Faunas from Lower and Middle Mississippian Strata in Southern Illinois, Circular 255, gratis, 1958.

Mineral Production in Illinois in 1957, Circular 257, gratis, 1958.

Fuels and Power in Manufacturing Industries, Circular 259, 1958.

Indiana

Order From:

Dept. of Conservation
Publications Section, Geological Survey
Indiana University
Bloomington, Ind.

Atlas of Mineral Resources of Indiana, Map No. 10, by William J. Wayne, 50¢, 1958.

Let's Look at Some Rocks, Circular No. 5, by William J. Wayne, 35¢, 1958.

Kansas

Order From:

State Geological Survey
University of Kansas
Lawrence, Kan.

Geology and Ground-Water Resources of Logan County, Kansas, Bulletin 129, by Carlton R. Johnson, $1, 1958.

Geology, Mineral Resources, and Ground-Water Resources of Elk County, Kansas, Vol. 14, **Part I** by John M. Jewett; **Part II** by Robert Kulstad, Norman Plummer, W. H. Schoewe, and E. D. Goebel; **Part III** by C. K. Bayne, 75¢, 1958.

Petrology of the Bliocene Pisolitic Limestone in the Great Plains, Bulletin 130, **Part 2**, by Ada Swineford, A. Byron Leonard, and John C. Frye, 25¢, 1958.

Flowage in Rock Salt at Lyons, Kansas, Bulletin 130, **Part 4**, by L. F. Dellwig, 25¢, 1958.

Ground-Water Levels in Observation Wells in Kansas, Bulletin 131, by V. C. Fishel and B. J. Mason, $1, 1958.

Maine

Order From:

State Geologist
Dept. of Economic Development
State Office Bldg.
Augusta, Maine

Maine Granite Quarries and Prospects, Minerals Resources Index No. 2, $1, 1958.

Maryland

Order From:

Dept. of Geology, Mines, and Water Resources
The Johns Hopkins University
Baltimore 18, Md.

Water Resources of Cecil, Kent, and Queen Annes Counties, Bulletin 21, by Robert M. Overbeck, T. H. Slaughter, and A. E. Hulme, $5, 1958.

Minnesota

Order From:

University of Minnesota Press
Minneapolis 14, Minn.

Geological Map of Minnesota, $1.

Maps and Diagrams of the Mesabi Range, $1.50.

Missouri

Order From:

Thomas R. Beveridge
State Geologist
Missouri Geological Survey and Water Resources
Box 250, Rolla, Mo.

Resistivity Surveys of Missouri Limonite Deposits, Report of Investigations No. 24, by T. Meidav, W. C. Hayes, and G. E. Heim, 25¢, 1958.

Publications of the Div. of Geological Survey and Water Resources, Dept. of Business and Administration, State of Missouri, gratis, 1958.

Montana

Order From:

Bureau of Mines & Geology
Room 203-B, Main Hall
Montana School of Mines
Butte, Mont.

Bibliography of Publications of the Montana Bureau of Mines and Geology, Information Circular No. 3, by Uuno M. Sahinen, gratis, 1957.

Progress Rport on Clays In Montana, Information Circular 23, by U. M. Sahinen, R. I. Smith, and D. C. Lawson, gratis, 1958.

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The Mineral Industry of Montana in 1957, Area Report B-63, available from the Bureau of Mines, Div. of Mineral Industries, Region I, P. Box 492, Albany, Ore.

Nevada

Order From:

Nevada Bureau of Mines
University of Nevada
Reno, Nev.

Iron Ore Deposits of West-Central Nevada, Bulletin No. 53, $1.50.

New Mexico

Order From:

New Mexico Bureau of Mines and
Mineral Resources
Campus Station
Socorro, N. M.

Geology of the Cerrillos Area, Sante Fe County, New Mexico, Bulletin 48, by A. E. Disbrow and W. C. Stoll, $3, 1957.

Geology of Dog Springs Quadrangle, New Mexico, Bulletin 58, by David B. Givens, $2, 1957.

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Topical Study of Lead-Zinc Gossans, Bulletin 46, by William C. Kelly, $1.50, 1958.

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Geology and Mineral Resources of Mesa Del Oro Quadrangle, Socorro and Valencia Counties, New Mexico, Bulletin 56, by H. L. Jicha, $2.50, 1958.

Wall-Rock Alteration in the Cochiti Mining District, New Mexico, Bulletin 59, by W. M. Bundy, $2, 1958.

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Geologic Map of Inscription Rock Fifteen-Minute Quadrangle, Map No. 4, by Clay T. Smith, 50¢, 1958.

Reconnaissance Geologic Map of Canon Largo, Thirty-Minute Quadrangle, Map No. 6, 50¢, 1958.

New Mexico Energy Resources Map, $1, 1958.

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Preliminary Geologic Map of the Northwestern Part of New Mexico, by C. H. Dane and G. O. Bachman, 75¢, 1958.

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New Mexico Business, monthly, $2 per year, 50¢ per issue, available from Bureau of Business Research, University of New Mexico, Albuquerque, N. M.

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North Dakota

Order From:

State Geologist
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Crabs from the Canonball Formation (Paleocene) of North Dakota, Miscellaneous Series No. 11, by F. D. Holland, Jr., 50¢, 1958.

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BOOKS (*Continued from page 265*)

29, by C. B. Folsom, Jr., S. B. Anderson, and Miller Hansen, $1, 1958.
Geologic Map of the Madison Group in the Bottineau Area of North Dakota, by S. B. Anderson and C. G. Carlson, $1, 1958.

U. S. Bureau of Mines

Copies sold through:

Superintendent of Documents
U. S. Government Printing Office
Washington 25, D. C.

RI 5400 Investigation of the Cuyuna Iron-Range Manganese Deposits, Crow Wing County, Minn., 40¢.
IC 7842 Injury Experience in the Quarrying industry, 1954, 30¢.
IC 7840 Permissible Mine Equipment Approved During the Calendar Years 1955-1956, 15¢.
IC 7859 Injury Experience in Coal Mining, 1953-1954, 50¢.
IC 7863 Stone Cutting and Polishing, 25¢.
Foreign Commerce Weekly, $4.50 a year, 10¢ per copy.
World Trade Information Service, Part I, economic reports, $6 a year, 10¢ a copy; **Part II,** operations reports, $6 a year, 10¢ a copy; **Part III,** $6 a year, 20¢ a copy.
Comprehensive Export Schedule, published annually, $6 a year, 10¢ a copy for single **Current Export Bulletins.**
Investment Opportunities Abroad, weekly bulletin available from Bureau of Foreign Commerce, U. S. Dept. of Commerce, Washington 25, D. C.
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Channels For Trading Abroad, 25¢.
A Directory of Foreign Development Organizations For Trade and Investment, 30¢.
A Guide To Foreign Business Directories, 45¢.
International Geophysical Year, Catalog No. 84-2.Sdoc. 124, 30¢.
46V Statistics of Higher Education: 1955-1956, Faculty, Students, and Degrees, Catalog No. FS 5.23:954-56, 60¢.
51V Structural and Igneous Geology of the La Sal Mountains, Catalog No. I19.16:294-I Utah.
30W Hot Laboratory Equipment, Catalog No. Y3.At.7:2L11/2, $2.50
36W Progress Report on Science Programs of the Federal Government, Catalog No. 85-2:S.rp.2498, 25¢.
21 Facts about Coal, Catalog No. I28.2:C63/6-/955, 25¢.

15Z Coke Plants in the United States As of December 31, 1957, Catalog No. I28.27:7861, 20¢.
RI 5435 The International Systems of Hard-Coal Classification and Their Application to American Coals, 20¢.
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IC 7844 Southeastern Alaska's Mineral Industry.
IC 7846 Chemical Solidification of Soil in Tunneling at a Minnesota Iron-Ore Mine.
IC 7848 Methods and Operations at the Yerington Copper Mine and Plant of the Anaconda Co., Weed Heights, Nev.
IC 7849 Mining Inclined Beds of Phosphate Rock, San Francisco Chemical Co., Mines, Rich County, Utah.
IC 7852 Fire-Protection System, Allen Coal Mine, Colorado Fuel and Iron Corp.
IC 7853 Coal-Mine Ventilation Without Doors to Control Main Air Currents.
IC 7854 Exploration, Development, and Costs of the Stormy Day Tungsten Mine, Pershing County, Nev.

ABSTRACTS

In This Issue: *The following abstracts of papers in this issue are reproduced for the convenience of members who wish to maintain a reference card file and for the use of librarians and abstracting services. At the end of each abstract is given the proper permanent reference to the paper for bibliography purposes.*

History and Geology of the Berkeley Pit

I. **Introduction** by *Charles G. Goddard, Jr.*—The Butte district has produced over $2,250,000,000 in copper, zinc, manganese, silver, and gold, unprecedented in the mining world. Exhaustion of the gold placers, located in 1864, was followed by Ag-Au mining from many of the vein outcrops. Montana was the leading silver-producing state in the 1880's. Deeper mining disclosed copper occurred in abundance. From this beginning Butte became the greatest copper-producing district in the world. During this first period to 1910, 6,600,000,000 lb of copper were produced. At the time of the initiation of the Kelley mine, another 6,600,000,000 lb of copper were added, and at the end of 1957, a total of 14,604,000,000 lb of copper were produced during Butte's 78-year history.

Deep levels continue to show the same type of copper mineralization that was found on high levels. This continuing resource and improvements in mining, mechanization, and efficiency allow Butte to continue as an important producer of mineral wealth. The Berkeley, planned to augment production of the deeper mines and the Kelley block cave mine, was made possible as the result of studies of old geologic records and incomplete sampling data. The existence of a supergene enriched sulfide orebody near enough to surface for open pit mining was proven by underground sampling crosscuts and extensive churn drilling from surface.

II. **Service Facilities for Mobile Equipment at Berkeley Pit Operations** by *Paul M. Young*—This part describes the planning and selection of the necessary service facilities for all of the mobile equipment used in the Berkeley Pit operation.

A list of the mobile equipment necessary for the pit operations is given, as well as a description of the various buildings required to house all of the facilities. The various shop areas are listed along with the tools and equipment for the various service operations. A description of the fuel storage and handling system as well as the lubrication service is included also.

III. **Berkeley Pit Crusher and Conveyor System** by *Frank Ralph*—This section of the paper pertaining to the Berkeley Pit operation covers the mechanical rock handling system from truck to railroad car, consisting of truck dumping facilities, a screening and crushing plant, five main conveyor belts, stockpiling equipment, storage bins, and railroad car loading facilities capable of handling 2000 tph of ore.

IV. **Layout and Mining of Berkeley Pit** by *Edward O. Bonner*—The introduction to this part covers the following information: start of open pit mining in the Butte district; a description of the methods used to verify the existence of a large low grade orebody; pit design including leach and waste disposal areas; long range planning for the pit; tonnage of ore, leach, and waste material that will be removed from the pit; and past and present production statistics. **Ref.: MINING ENGINEERING, March 1959, pp. 289-300.**

Tungsten in Searles Lake by *L. G. Carpenter* and *D. E. Garrett* (TP 4788I)—Probably the largest single tungsten deposit in the U. S. is one that has yet to produce any tungsten; it is not even listed in tables showing U. S. reserves. This deposit is at Searles Lake, Calif., where the brine contains about 70 ppm WO_3. Small though this concentration is, the amount of brine in the lake is so great that it is estimated at 170 million lb of WO_3 (8.5 million units) equalling the total of all other known reserves in the U. S. An unexploited reserve promises more tungsten than the total contained in all other known sources in the U. S. The problem now is commercial recovery. **Ref.: (MINING ENGINEERING, March 1959) AIME Trans., 1959, vol. 214, p. 301.**

Mineral Dressing Fundamentals Applied to the Fine Coal Problem by *M. C. Chang* and *John Dasher* (TP 4795F)—The Crucible mine, operating on Pittsburgh seam coal, is located at Crucible, Pa. The rated production capacity of the mine is about 5000 tpd. The washing plant, built by Roberts and Schaefer in 1943, has a rated capacity of about 400 tph, and uses hydroseparator boxes and hydrotator units to wash the coarse coal (4 in. by 5/16 in.) and the fine coal (5/16 in. by 0), respectively. This paper discusses the solution to the problems associated with the fine coal section of the washing plant. **Ref.: (MINING ENGINEERING, March 1959) AIME Trans., 1959, vol. 214, p. 304.**

A Decade of Development in Surveying by *Robert W. Baldwin* (TP 4793L)—*Overvoltage*, as applied to geophysical exploration, is the phenomenon whereby a current into the earth sets up secondary voltages which decay when the current is interrupted. These secondary effects may be measured with the aid of pick-up electrodes. The term *induced polarization* has been frequently employed to describe this same phenomenon. In its own operations Newmont Exploration Ltd. commonly uses the word *pulse*.

The basis of the overvoltage or induced polarization method as a prospecting device is that metallic particles, in particular sulphides, give a high response, whereas barren rock, with certain exceptions, gives a low response. The overvoltage method has been tried in the search for a number of types of mineral occurrence, but has found its most useful application in outlining widespread disseminated mineralization such as associated with porphyry coppers. **Ref.: (MINING ENGINEERING, March 1959) AIME Trans., 1959, vol. 214, p. 307.**

Processing California Bastnasite Ore by *C. J. Baroch, M. Smutz,* and *E. H. Olson* (TP 4794L)—In 1949, a large radioactive rare earth orebody was discovered in the Mountain Pass district of San Bernardino County, Calif. In 1950 Molybdenum Corp. of America purchased some claims at the deposit. By floating the barite and depressing the bastnasite Molybdenum Corp. produced a concentrate that contained 60 to 70 pct rare earth oxides. But this operation lost over 25 pct of the rare earths. At the present time most of the rare earths in the concentrates are converted to rare earth chlorides, which are either fed to ion exchange columns for separation of the individual rare earths or reduced to misch metal. **Ref.: (MINING ENGINEERING, March 1959) AIME Trans., 1959, vol. 214, p. 315.**

Interaction of Minerals with Gases and Reagents in Flotation by *Igor Plaksin* (TP 4790B)—The interaction of sulfide minerals and native metals with reagents in flotation is largely determined by the changes in particle surfaces resulting from the action of the medium and dissolved gases upon them. A number of early studies were devoted to the selective action of various gases upon minerals. One study paid attention to oxygen as the most active gas affecting the flotation properties of sulfide minerals. In the present study it was concluded that: 1) Gases play an important chemical role in flotation. The behavior of minerals in particular depends on the concentration of oxygen in solution. 2) The effect of oxygen is not limited to sulfide minerals, but surprisingly enters into the behavior of non-sulfides. 3) The crystal structure of minerals, sulfides, and non-sulfides alike, is a major factor in their response to variations in oxygen level. **Ref.: (MINING ENGINEERING, March 1959) AIME Trans., 1959, vol. 214, p. 319.**

Experiments in Concentrating Iron Ore from the Pea Ridge Deposit, Missouri, by *M. M. Fine* and *D. W. Frommer* (TP 4791B)—Mineral dressing research showed that iron concentrates of commercial quality could be produced from the Pea Ridge deposit near Sullivan, Mo. Magnetic separation and flotation, on a laboratory scale, yielded concentrates ranging in iron content from 60.0 to 71.4 pct at recoveries of 89.9 to 95.0 pct. **Ref.: (MINING ENGINEERING, March 1959) AIME Trans., 1959, vol. 214, p. 325.**

Permissible-Type Dust Counter for Coal Mines by *A. L. Thomas, Jr.* and *Sabert Oglesby, Jr.* (TP 4797F)—One of the primary purposes of sampling air-borne dusts in the mine is to maintain the necessary information concerning the nature and magnitude of dust exposure of personnel. The subject has been of interest for a number of years in the mining industry, but indications are that the interest is increasing as a result of the continued and increased awareness of industrial health problems associated with air-borne dusts. **Ref.: (MINING ENGINEERING, March 1959) Trans., 1959, vol. 214, p. 328.**

MANUFACTURERS NEWS

NEWS / EQUIPMENT / CATALOGS

Biggest Cat Grader

Largest and most powerful motor grader ever manufactured by *Caterpillar Tractor Co.*, the new No. 14 grader is rated at 150 hp and weighs more than 29,000 lb. The 12-ft moldboard is 27 in. high. Transmission

provides six forward and two reverse speeds, with forward speeds ranging from 2.6 to 21.6 mph. Top efficiency is obtained through balanced relationship of size, weight, and horsepower. Power assists are standard. Dry type air cleaner cuts servicing time and costs. **Circle No. 1.**

Primary Screen

Derrick Mfg. Co. offers the high frequency Derrick BF primary screen, designed for coarse separation (20 mesh, 4-in. opening). Screen operates at 1800 rpm and is designed to prevent blinding. Screen sizes range from 4x8 ft to 5x12 ft. **Circle No. 2.**

Hydra-Lift

Pitman Mfg. Co.'s new model 50 Hydra-Lift lifts up to 5000 lb, requires only 22 in. behind a truck cab, is completely hydraulic, and boom rotates a full 360°. As shown by dotted lines on photo, boom can be carried over truck or bed (upper portion of telescoping boom not shown). Three model choices: fixed length boom, 12 ft long; manual telescope boom, from 12 to 16 to 20 ft; hydraulic telescope boom, from 12 to 20 ft. **Circle No. 3.**

End Dump

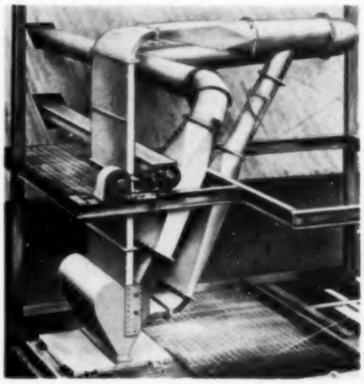
Model W-2800 by *Oshkosh Motor Truck Inc.* is a 30-ton end dump with 4-wheel drive. Features include hinged instrument panel, independent engine firewall, Cummins or Hall-Scott engine, Vickers power steering, 68-ft turning radius with standard wheel base. **Circle No. 4.**

Slurry Valve

Flow through the *Clarkson Co.* model C slurry valve is controlled by a rubber venturi restriction, air or hydraulically operated. Two-piece housing is jacketed steel. Rubber sleeve liner can be replaced without dismantling. Full diameter sizes are 1½, 2, 3, and 4 in. **Circle No. 5.**

Classifiers

Gravitational-inertial classifiers newly introduced by *Buell Eng. Co.* are compact units of 100-lb to 100-ton per hr capacity, designed for operation in the 200 to 50-mesh



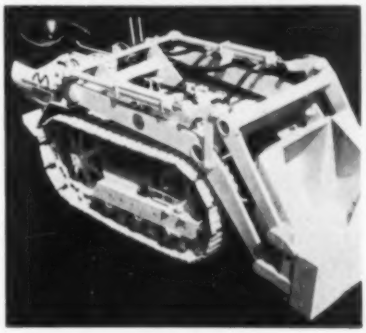
range. No moving parts, little maintenance, very low power requirements. **Circle No. 6.**

Package Disc Filter

Dorr-Oliver Inc. announces it has a new package-type American disc filter for dewatering coal and other fine flotation concentrates. Receiver, vacuum and filtrate pumps, motors, and controls are arranged on a skid platform according to individual needs. Special receiver allows mounting filtrate pump in line with filter. Unit retains design of standard American disc filter, but features fabricated pipe center shaft and easily changed disc sectors which are covered with 60-mesh stainless steel wire mesh. Operating with a −28 mesh coal feed containing 30 pct solids at 50 psi per hr, units will deliver a 75-pct solids cake at capacities ranging 4300 to 15,000 lb per hr. **Circle No. 7.**

Air-Powered Loader

A new front-end air-powered loader and dozer by *Machinery Center Inc.* will operate under headings as low as 4 ft and reach up to 6 ft dumping height. A 15-hp reversible air motor powers through direct drive 3-speed transmission with travel speeds to 5 mph. Steering clutches and bucket are hydraulically controlled. Four models: ⅜, ⅝, ⅝, and 1 cu yd. Air requirement is 315 cfm at 80 psig. **Circle No. 8.**



Short-Range Geodimeter

Svenska AB Gasaccumulator has introduced the model 4 geodimeter for accurate measurement of unknown distances up to 3 miles, even in rough terrain. Three frequencies are used to ensure accuracy of 0.04 ft on any line from 50 ft. Modulated light beam is reflected by pre-established and unmanned reflectors stationed at the unknown points. Geodimeter weighs only 35 lb, measures 11x13x12 in., can also be used underground. Unit is distributed by *Berg, Hedstrom & Co. Inc.* **Circle No. 9.**

Vibrating Screen

The Screen-Master, developed by *Universal Eng. Corp.*, is a horizontal vibrating screen equipped with phasing bars, air springs, and rubber bushings for higher capacity and smoother action. Models are available with 2, 2½, or 3 decks in sizes from 3 x 8 ft to 4 x 12 ft. **Circle No. 10.**

Deep-Hole Drill

Ingersoll-Rand has introduced a deep-hole hammer drill, model D40, utilizing both conventional and reverse rotation. Selector incorporated into the backhead permits instant rotation change for convenience in coupling or uncoupling steel. D40 has a 4-in. bore and handles bits up to 3-in. diam. Drill is used with Ingersoll-Rand FM4 drill guide, which allows application to a variety of mountings. **Circle No. 11.**

(*Continued on page 268*)

Manufacturers' News

(*Continued from page 267*)

Multi-Purpose Tractor-Truck

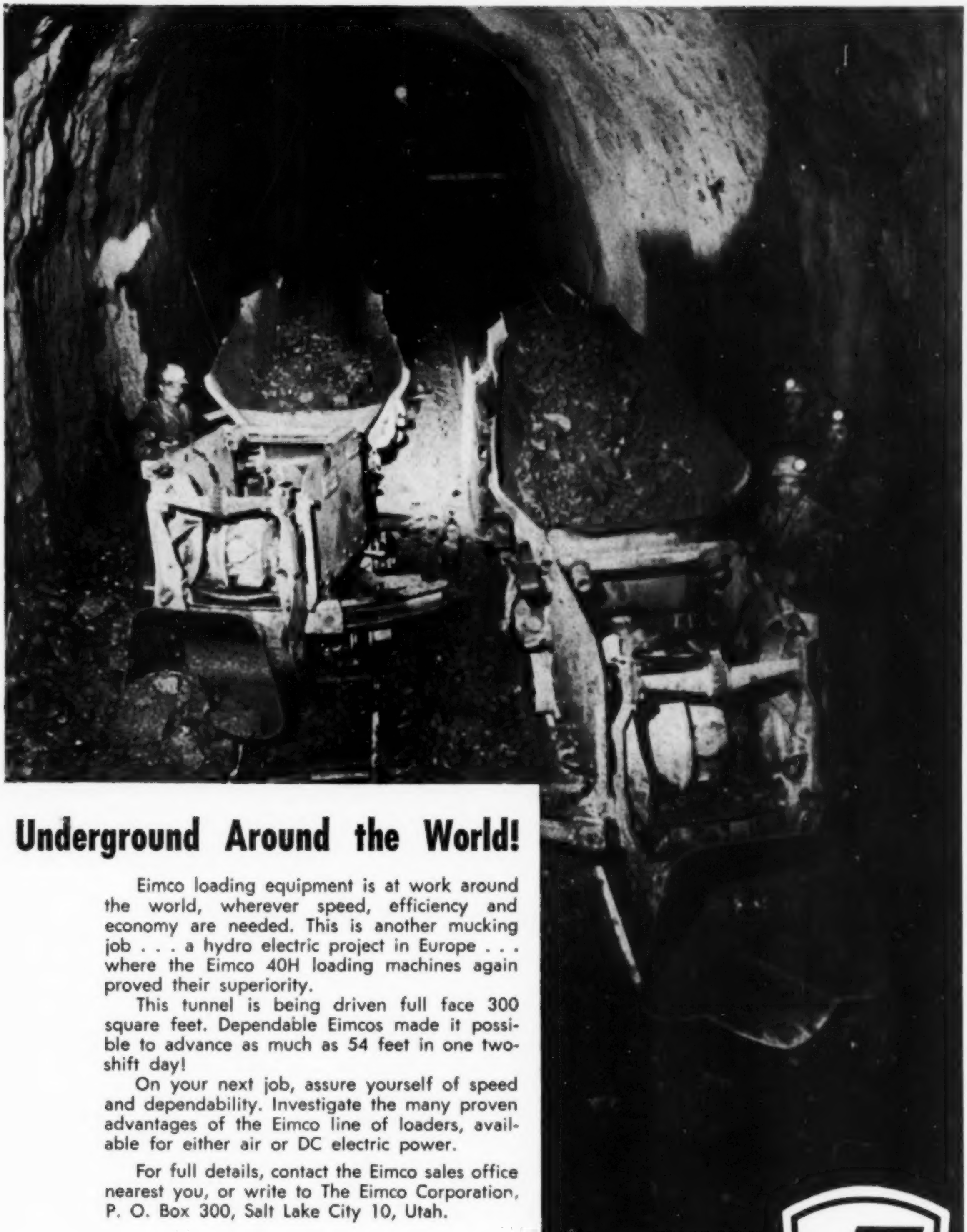
The Mercedes-Benz Unimog is a diesel-powered combination tractor capable of pulling 20 times its own weight or carrying a 1¼-ton payload. Truck travels easily over rough terrain, can accomplish a 60° climb or a 38° traverse with full payload. The Unimog uses a maximum of 1½ gal of diesel fuel per hr, for either field or stationary work. Unit is sold by *Curtiss-Wright Corp.*, South Bend Div. **Circle No. 12.**

Uphill Conveyor

Carrier Conveyor Corp. has developed a new mechanical vibrating conveyor for moving solid granular material, sand, and tramp iron uphill at inclines of 5 to 25°. Trough *steps* catch material on the downstroke to prevent it from slipping backward. Conveyor features Natural-Frequency drive and is built to resist heat, abrasion, and impact. **Circle No. 13.**

Rubber-Lined Pump

Allis-Chalmers has extended the capacity range of its rubber-lined pumps by adding a new 14 x 12-in. unit. The pump is designed for heads up to 140 ft at 870 rpm and capacities to 8000 gpm. Overhead motor mounting conserves floor space. V-belt drive is standard Texrope. **Circle No. 14.**



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BRANCHES AND DEALERS IN PRINCIPAL CITIES THROUGHOUT THE WORLD B-391

Depletion Tax Law Probed by Treasury

Treasury Secretary R. B. Anderson has asked Congress to call a halt to the practice of allowing mining firms to base their tax deductions for depletion on the price of the fabricated product instead of on the worth of the raw ore. Mr. Anderson said the Government was losing revenue because of liberal court interpretations of the old law and called for a new statute which would prevent depletion deductions on this "inflated scale."

Copper Price Up, May Go Higher

The large primary copper producers have yielded to the pressure of higher world prices and raised their quotation from 29¢ a pound to 30¢, the first upstep since late October 1958. Consumption lately has been on the increase and inventories are being replenished by fabricators. In addition to these price-firming factors there is fear in the market that strikes may flare after the current labor contracts run out on June 30. The strike possibility is expected to have increased importance as that date grows near and may encourage a further increase in the selling price of the red metal. The custom smelters have already boosted their price to 31¢, there is popular unrest near certain of the African producing facilities, and a strike goes on at Cerro de Pasco in Peru—all factors adding pressure for a still higher rise in the price quoted by the primary producers.

105 Million Tons of Steel in 1959?

Output of 105 to 115 million tons of ingot steel this year was predicted recently by M. J. Aurelius, vice president, U. S. Steel Corp. Mr. Aurelius believes the first six months of the year will bring production approaching 60 million ingot tons, a 55 pct upstep over the same period in 1958. According to the American Iron & Steel Institute, January ingot production, 9.31 million net tons, was the largest since mid-1957. Tonnage is now approaching 87.5 pct of capacity.

New Iron Mine, Steel Plants

Lowphos Ore Ltd. at Moose Mountain, Ont., will begin mine and plant production of high grade iron concentrates on April 1. The operation, originally scheduled to open last spring, was delayed by reduced demand. At full operation output will reach about 550,000 tons annually. . . . An experimental H-iron pilot facility is planned by Bethlehem Pacific Coast Steel Corp.'s Los Angeles plant. The direct reduction plant will comply with air pollution regulations and will be first of its type on the West Coast. . . . A group of Canadian companies has intentions of building a steel plant that will employ the Strategic-Udy process, which permits bypassing the blast furnace and open hearth methods in standard steel making. Koppers Co. Inc., potential builder of the facility, has not identified the group concerned nor the possible size of the plant. Cost of plants using the Strategic-Udy process is $30 to $50 per yearly ingot ton of capacity.

Bigger Blast Furnaces for USSR

The Soviet Union has voiced plans for several huge steel-making furnaces, some of 2615-cu yd capacity and one of 2985 cu yd, that are intended to turn out steel at 5 pct less than normal cost. Design work is claimed underway and foundations are to be started next year.

AEC May Stretch Out Uranium Purchases

The Atomic Energy Commission, expecting delivery in the next three years of more uranium than it will be able to use, has started discussing with industry the possibility of stretching out its uranium purchases by cutting back deliveries over the short term and adding the undelivered quantities of metal to purchase commitments in the future. The move would affect the large producers and not the smaller operators. Industry is leery because of possible heavy profit loss that might stem from climbing costs and interest payments on their loans. Producers also note that large U. S. contracts for material have been negotiated with foreign producers who, on the average, have been receiving more per pound for their concentrate from the U. S. Government than have domestic producers.

Predict Bigger Consumption of Nickel in 1959

A Department of Commerce agency predicts nickel consumption this year will climb to 95,000 to 100,000 tons, well above the 80,000 tons in 1958. The prediction, however, is predicated on sustained improvement in general industry and on business in the automobile and steel industries in particular. Defense needs were estimated by the survey to remain about the same as last year.

Mercury Industry Termed on Unsteady Ground

The end of the Government price support program on December 31 last year, coupled with heavy overseas competition and lower world prices, could undercut all the forward steps of the domestic mercury industry made with Government help, the Justice Department told Congress in a Defense Production Act report. The Department termed the future success of the industry "problematical" because "deposits are still small, irregular, and scattered" and "generally poorer than those in some foreign countries," while "production costs remain high in comparison."

Lead and Zinc Both Decline to 11c per Lb

Custom smelters of lead and zinc instituted a series of price cuts that brought both metals to the 11¢-per-lb level almost simultaneously. The reduction in the price of lead marked the third slash since January 21 bringing the price down 2¢ since that time and virtually wiping out all the price gains that followed the setting of import quotas last fall. The ½¢ chop on the zinc quotation was the first change since the price rose to 11½¢ last November. It came as a surprise since good quantities of slab metal have recently been delivered to steel makers for galvanizing.

USSR Short of Aluminum, Russian Declares

A. B. Aristov, member of the Soviet Presidium, in early February said the USSR has an inadequate supply of aluminum and the situation will probably last for some years. In a statement before a Communist party congress Mr. Aristov declared the problem so serious that the original Seven Year Plan goal of boosting output by 1965 to 2.8 to 3 times its 1958 total would be too small. He called for a still greater increase in production. Only a short time ago the Russians were actively supplying world markets with substantial quantities of aluminum in a program which domestic producers scored as "dumping."

heart of a Giant

The head and shaft of 1'-3" Traylor Type TY Gyratory Reduction Crusher suspended next to the massive head and shaft of a 60" Traylor Type TC Bulldog Gyratory Crusher. One weighs 480 pounds, the other 140,000 pounds.

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Traylor Engineering is a specialist in designing and building primary gyratory crushers. The mammoth 60" crusher is capable of delivering 32,000 tons of product per eight-hour day. Traylor Gyratory Crushers feature original non-choking, self-tightening bell head and curved concaves. Write for Bulletin No. 1126 today.

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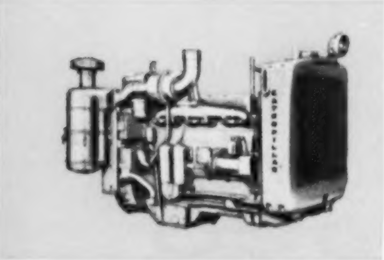

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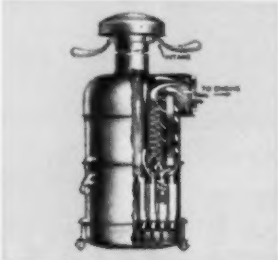

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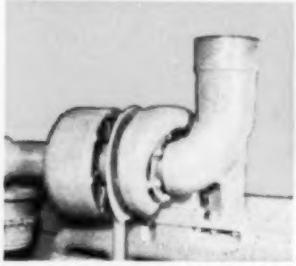

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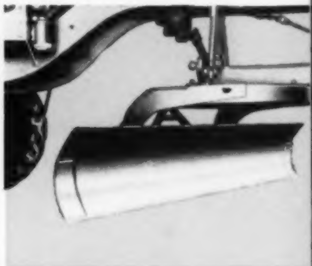

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Uranium Review, Outlook*

*Excerpts from remarks by Jesse C. Johnson, Director of the Div. of Raw Materials, U. S. Atomic Energy Commission in Denver, Feb. 6, 1959.

The U. S. uranium industry in the past ten years has grown from practically nothing to a position of major importance. In 1958 more than 5 million tons of uranium ore were mined in our western states. This ore was processed in local mills which produced concentrates containing about 12,500 tons of U_3O_8. These concentrates were sold to the Atomic Energy Commission for approximately $238 million, exceeding by a wide margin the combined value of domestic lead and zinc production.

Domestic uranium production nearly doubled from 1956 to 1958 and 1959 will show another large increase. Concentrate production in 1959 will approximate 18,000 tons of U_3O_8 having a value of more than $300 million—which may be about the level of production through 1966. For a new metal, rushed into production to meet a military requirement, uranium will have the enviable record of rapid expansion and then stability.

However, a rapidly expanding mining industry provides more new opportunities than one operating at a stable rate—even though a high rate. The uranium producers have only a government market—not by reason of an imposed monopoly, but because, as yet, there is no other significant market. Consequently, the AEC's recent procurement policies, which have had the effect of limiting further expansion of domestic uranium production, have been the subject of much discussion. Foreign uranium contracts, especially, have been criticized.

For a better understanding of the current problems, and for answers to questions now being raised, it is necessary to look at the conditions which existed when the search for uranium began and when the various foreign commitments were made. With this in mind, let us review briefly.

In 1948 nearly all of this country's uranium supply came from two small mines—the Eldorado on Great Bear Lake in Arctic Canada, and the Shinkolobwe in Belgian Congo. The production rates of these mines were limited and we never could safely count on ore reserves for more than a few years ahead. Today neither mine is an important factor in our uranium supply. Had no new sources of production been developed, essential military requirements could not have been met. Furthermore, there would have been no basis, either for this country or for Europe, to have undertaken extensive development of atomic energy for industrial use.

In 1948 we knew of only three large sources of uranium-bearing materials, all low-grade—the South African gold ores and our domestic shale and phosphate deposits. In the case of the South African gold tailings, the problem was to develop a process which would recover economically from ¼ to ½ lb of U_3O_8 from a ton of material. Our domestic shales would provide less than 1/10 lb of oxide per ton. Our phosphate industry promised only limited production of byproduct uranium. Byproduct recovery from South African gold tailings appeared to be the most promising. However, the AEC undertook extensive research and development for recovery of uranium from all three sources—gold tailings, shale, and phosphate.

Arrangements were completed at the end of 1950 for a major South African production program. Except for three small phosphate byproduct units, the domestic shale and phosphate programs did not get beyond the research and development stage.

Aside from these three low-grade sources, additional large-scale uranium production depended upon new discoveries. In January 1948, developed and partially developed ore reserves of the Colorado Plateau were estimated at approximately 1 million tons, containing less than 2500 tons of recoverable uranium oxide. These were the only known deposits of commercial grade uranium ore in the U. S. Some might say that this was an incomplete or pessimistic estimate, or that more weight should have been given to potentialities. It should be pointed out that five years later, at the beginning of 1953, estimated ore reserves were only 3 million tons containing about 6000 tons of U_3O_8. A year later they were still less than 5 million tons and at the beginning of 1955, seven years after the domestic program was established, they were approximately 10 million tons, less than two years' supply at today's mining rate.

Obviously, during these years we neither could depend upon new discoveries, which are unpredictable, nor delay getting production from every available known source. These were the years in which foreign commitments were made. From 1948 to 1955 military requirements for uranium greatly increased.

It was not until 1954, six years after the AEC domestic uranium program was announced, that private industry really became active. During the earlier years, the only major exploration drilling programs were those of the AEC. In the eight years, 1947 to 1954, inclusive, the Commission spent $46 million for geological investigations, drilling programs, and the development and use of radiometric logging and airborne radiometric surveys. During this period prospectors and small mining organizations were the most active. The discoveries made by these pioneers and the geological information and exploration methods developed by the Commission stimulated the large-scale private exploration that followed. Prior to 1953, private drilling was less than 1 million ft per year and most of this was development drilling, extending the limits of known ore bodies, rather than exploration for new deposits. In 1957, private drilling reached a peak of about 9 million ft.

The AEC also had to develop new and better processes for milling uranium ores. Its process development program is responsible for the basic processes used today in all of our uranium mills. Metallurgical recovery of about 90 pct has become standard practice. In 1948, the mills recovered only about 70 pct of the uranium content of the ores.

These metallurgical improvements, together with larger milling operations, have been responsible for

lower milling costs and the lower prices now paid by the AEC for concentrate. The Commission is reaping benefit from its expenditure of $26 million for process development. There has been no reduction in the price paid to the miner for his ore.

By the beginning of 1955 the outlook for uranium supplies had greatly improved. The Canadian Blind River field, which had been discovered in mid-1953, already had developed a large production potential. Domestic production was increasing rapidly although ore reserves were much too small in relation to this country's current and future requirements. Consideration of these factors was the basis for limiting Canadian purchases and continuing to provide a market for new domestic discoveries.

As a result of our conversations with Canadian officials, Canada in August 1955, announced that no additional milling contracts would be negotiated except for projects that could qualify for a contract by Mar. 31, 1956. This action had the effect of limiting contracts to mines with development well advanced. As a matter of fact, all mines able to qualify were discovered prior to 1955. Future uranium discoveries became ineligible for a Canadian government milling contract.

The U. S. on the other hand, continued to make contracts for additional domestic concentrate production. Furthermore, in May 1956, the AEC announced a domestic concentrate buying program extending through 1966. This action was based upon a review by the Commission of its projected requirements and purchase contracts, and upon the outlook for additional domestic production that might result from new discoveries.

The extensive exploration which followed that announcement was most successful. Private exploration drilling, which was estimated at 5½ million ft in 1955 was in excess of 9 million ft in 1957. Ore reserves, which were estimated at 30 million tons at the beginning of 1956, now are in excess of 80 million tons. Therefore, nearly 70 pct of today's reserves were developed in the last three years; more than 80 pct are in entirely new districts—districts in which the first discovery was made less than ten years ago.

With greatly increased ore reserves, rapidly expanding production and the probability of a continued high discovery rate, limits had to be placed on the 1962 to 1966 procurement program to avoid serious overproduction. These limits had to be such that the Commission's annual and total uranium commitments could be estimated with a reasonable degree of accuracy. Deliveries under the AEC's present domestic and foreign contracts, including domestic contracts under negotiation, will reach their highest level in the next three years. If there should be a need to expand production, this can be done from existing sources.

Much of the recently developed ore is in the form of large deposits which can support a high rate of production. Many of the deposits require relatively large-scale operations to be economic because of the heavy preproduction expenditures. Under a program that provided an adequate market for production from newly developed deposits of that type, we would be faced with the prospect of continued rapidly expanding production.

These were considerations which led to the Nov. 24, 1958, announcement. By the issuance of that announcement the AEC withdrew the 1956 domestic 1962 to 1966 program with respect to concentrate derived from ore reserves developed after November 1958. Concentrate produced from ore developed prior to that date may be sold to the AEC under negotiated contracts which will provide delivery rates designed to support a reasonable scale of operations. The previously announced price of $8 per lb of U_3O_8 in an acceptable grade of concentrate will remain in effect.

The negotiated contracts also will contain provisions designed to provide equitable treatment for independent miners, including a fair share of available mill capacity. During the period of uranium shortage a market generally was available for all acceptable ore produced or additional milling capacity was authorized. Under a limited procurement program, it may be necessary to allocate production rates to the various mines and to assign the production to specific mills. To the extent possible, mines and mills should work out their own arrangements within production limits established by the AEC.

The AEC's revised procurement program should provide domestic producers a market of $250 to $300 million a year through 1966. Domestic producers also may make commercial sales to licensed buyers.

What will be the effect of this revised program on exploration? Will domestic reserves be substantially exhausted by the end of 1966? These are questions frequently asked.

Private expenditures for exploration are based upon current and anticipated markets. From now on exploration, to a large extent, must be based upon the anticipated market—the market that will develop with the growth of industrial atomic power.

Many uranium producers already are planning for the industrial market. At least some of them can be expected to carry on long-range exploration programs. However, the general level of activity may be greatly reduced until the industrial demand for uranium can be timed and measured more accurately than at present. It should be before 1966.

As for rapid depletion of existing ore reserves, there is a good chance that reserves may be maintained near present levels for some time by extending the limits of known orebodies and producing areas. Geologists estimate that work of this type, without the discovery of new fields, may add 50 pct or more to present ore reserves. If some long-range exploration and development programs are continued, U. S. ore reserve position should remain satisfactory through 1966.

It is interesting to study the ore reserve figures for 1958, a year in which there was a substantial drop in prospecting and exploration. At mid-year it looked as if newly developed ore would offset the ore mined, and that the reserves at the end of the year would be about the same as at the beginning. Preliminary figures now indicate that ore reserves during the year increased by about 4½ million tons after mining in excess of 5 million tons, so that nearly 10 million tons of ore were developed. The knowledge and experience gained in exploring for uranium, and the successful results achieved, indicate that the undiscovered U. S. uranium resources are far greater than those already developed.

Domestic industry has achieved world leadership in the production of uranium. It should be able to maintain that position with the transition from a military to an industrial market.

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The following list of papers (from the 1959 San Francisco Annual Meeting) will be available until Jan. 1, 1960. Coupons received with the 1959 dues bills and those distributed at the 1959 Annual Meeting will also expire on this date. Purchased coupon books will be honored on any future date. A new listing of available papers will appear in a forthcoming issue. It will include additional papers presented at the 1959 Annual Meeting (San Francisco) and at other SME meetings throughout the year. Preprints may be obtained (upon presentation of properly filled out coupon) from SME Headquarters, 29 W. 39th St., New York 18, N. Y. Coupon books may be obtained from SME for $5 a book (10 coupons) for members or $10 a book for nonmembers. Each coupon entitles the purchaser to one preprint.

59H1—Water Law and Its Significance to the Mining Industry by Wells A. Hutchins.
59H2—Relation of Land Subsidence to Groundwater Withdrawals in the Upper Gulf Coast Region, Texas by Leonard A. Wood and A. G. Winslow.
59F3—Hydraulic Mining of Gilsonite and Its Application to Coal Mining by J. H. Baker.
59H4—Recovery of Phosphates by In Situ Fluid Mining by Sylvain J. Pirson.
59F5—Coal Cleaning Plant Design for Minimum Operating Labor by Wm. M. Bertholf and John D. Price.
59F6—Coke Combustibility: A Neglected Characteristic by J. D. Price.
59B7—Union Carbide's Uranium Operation at Maybell, Colorado by K. W. Lertz and F. T. Temple.
59H8—A Rapid Method for Estimating Alumina in Feldspathic Sands by Hugh H. Bein.
59H9—Water Laws as Related to Dredging in Idaho by Robert A. Lothrop, Richard B. Porter, and Robert P. Porter.
58B10—Separation and Washing of Alumina Process Residue by Morton Handelman.
59B11—Feed Preparation and Froth Modification for Fatty Acid Flotation by Carl C. Martin and Burt C. Mariacher.
59B12—Design Requirements for Tailing Disposal in the Southwest by E. Vern Given.
59B13—Semi-Dome Shaped Buildings for Bulk Storage by Edward E. Ives and William L. Payne.
59H14—Flow of Limestone and Clay Slurries in Pipelines by Ross W. Smith.
59H15—CO_2 Gas as a Cement Slurry Thinner by Duncan Williams and H. Potter.
59H16—Potash in Saskatchewan by Marion A. Goudie.
59B17—Pebble Milling Practice in the Reduction Works of the Gold Mines of Union Corp., Ltd. by O. A. E. Jackson.
59L18—The Bonanza Project, Bear Creek Mining Co. by Douglas R. Cook.
59J19—Preregistration Counseling for Mineral Industries Students at Penn State by John J. Schantz, Jr.
59H20—Colemanite as an Important Source of Borates by William T. Griswold.
59K21—Trends in Real Prices of Representative Mineral Commodities 1890-1957 by Charles W. Merrill.
59K22—Realignment of the Paley Commission Predictions Over the Next Five Years by S. G. Lasky.
59K23—Industrial Minerals 1950-1958-1975 With Special Emphasis on Fluorspar by Raymond B. Ladoo.
59F24—Removal of Sulfur Dioxide from Flue Gases at Elevated Temperatures by Daniel Bienstock and J. H. Field.
59B25—Ferrograde Concentrates from Arkansas Manganiferous Limestone by Morris M. Fine.
59F26—Are Coal-Mine Employees and Dollars Protected from Fire as well as Other Industrial Employees and Dollars by R. Ward Stahl.
59AU27—Ground Movement and Subsidence from Block Caving at Miami Mine by J. Fletcher.
59B28—Leaching, Ion Exchange and Precipitation, Blind River Uranium Ore by R. P. Ehrlich, A. G. Roach and K. D. Hester.
59AO29—Firing Fertilizer for Fragmentation by John R. Knudson.
59F30—The Integration of Coal Characteristics with the Design of Large Pulverized Coal Steam Generating Units by Douglas O. Hubert.
59K31—Light Metals—Prediction and Performance by Walter L. Rice.
59B32—Confirmation of the Third Theory by F. C. Bond.
59B33—Non-sulfide Flotation With Fatty Acid and Petroleum Sulfonate Type Promoters by Stuart A. Falconer.
59F34—A Laboratory Investigation of Flocculation As A Means of Improving Filtration of Coal Slurry by M. R. Geer, P. Jacobson, and H. Yancey.
59B35—The Gyratory Ball Mill, Its Principle of Operation and Performance by A. W. Fahrenwald.
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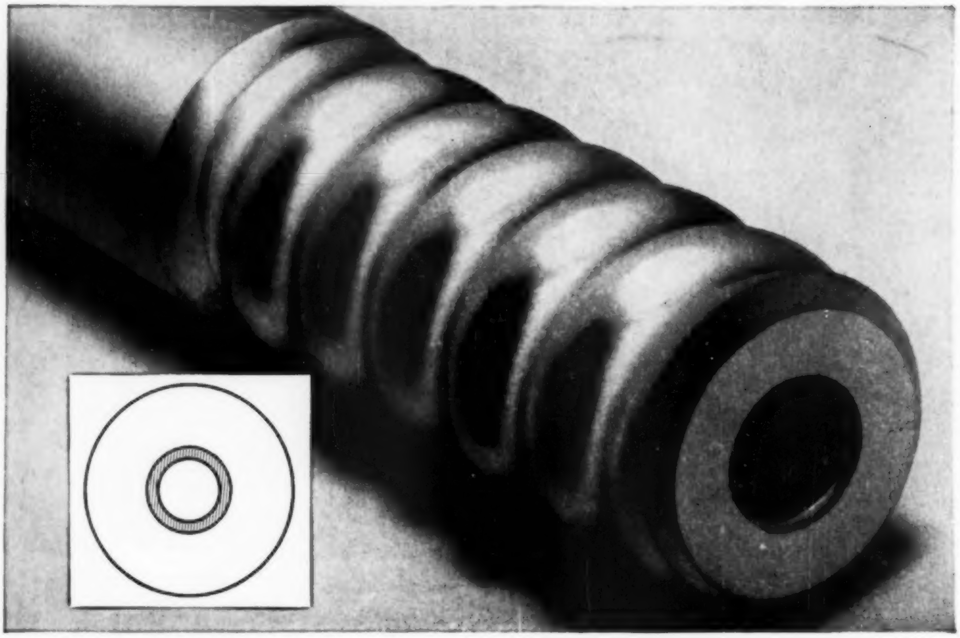

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MEETINGS, MEMBERSHIP, AND SERVICE

It is appropriate that *Drift* in this issue, which features J. W. Woomer, new SME President, should emphasize service and membership. Expanding existing services offered by the Society to its members and intensified membership campaigns will be major SME concerns throughout the year.

A professional organization based upon individual members is strong and grows only in proportion to those individual members. The expansion in 1959 of SME's Preprint Program is one example of increasing service to members. A second—and greater—increase in service comes through the Society's meeting on all levels: Local Section, regional, and national.

An approach to programming developed by the Colorado MBD Subsection and its committee under the direction of S. Power Warren may be one method for arranging a meeting with the broadest interest to members. Before beginning to draft a preliminary technical program for the Subsection's 1959 meeting in Colorado Springs, Mr. Warren set out to contact as many Subsection members and professional men in the area as he could, either in person or by mail or phone. In addition, he visited a number of plants in the surrounding area in order to obtain first hand knowledge of the type of problems and questions the operators and engineers would like to have discussed.

By the time a first *progress report* was made to the Subsection in August 1958, Mr. Warren was able to offer the broad topic outlines for a technical session. Most of those contacted also had suggestions for character of the meeting. Most suggested that there be short provocative papers, generally on one subject, followed by ample time for discussion from the floor. It was deemed necessary that at least statistics, flowsheets, and diagrams be preprinted and ready for distribution before each session. During each session, the chairmen should adhere to a rigid time schedule for presentation of each paper but the discussion time should be flexible. Ideas tentatively outlined in August had become firm as to general topics by the time Mr. Warren issued his second *progress report*.

In the second *progress report*, sent to over 100 operators who might attend the meeting, Mr. Warren outlined the breakdown of each topic. He also indicated, by a number of direct quotes, the reasons for choice of each topic.

Once the broad outlines of the program had taken shape, Mr. Warren and his committee set about lining up specific papers and authors. To cover the whole field of interest, members from plants treating different ores have been asked to cooperate in preparing a joint paper under a group chairman. Each group is deciding how the data is to be assembled and who is to present the paper. The program committee is coordinating the efforts of the groups and following progress closely in order to have material ready for preprinting prior to the April 18 meeting.

The group approach to meeting papers is an interesting one from a Society viewpoint. Not only does it help Section or regional members to become better acquainted, but it also introduces the professional organization (SME of AIME) to nonmembers who are professionally qualified for membership. The program approach of Colorado MBD is a good one and well worth the attention of other SME groups—*Rixford A. Beals.*

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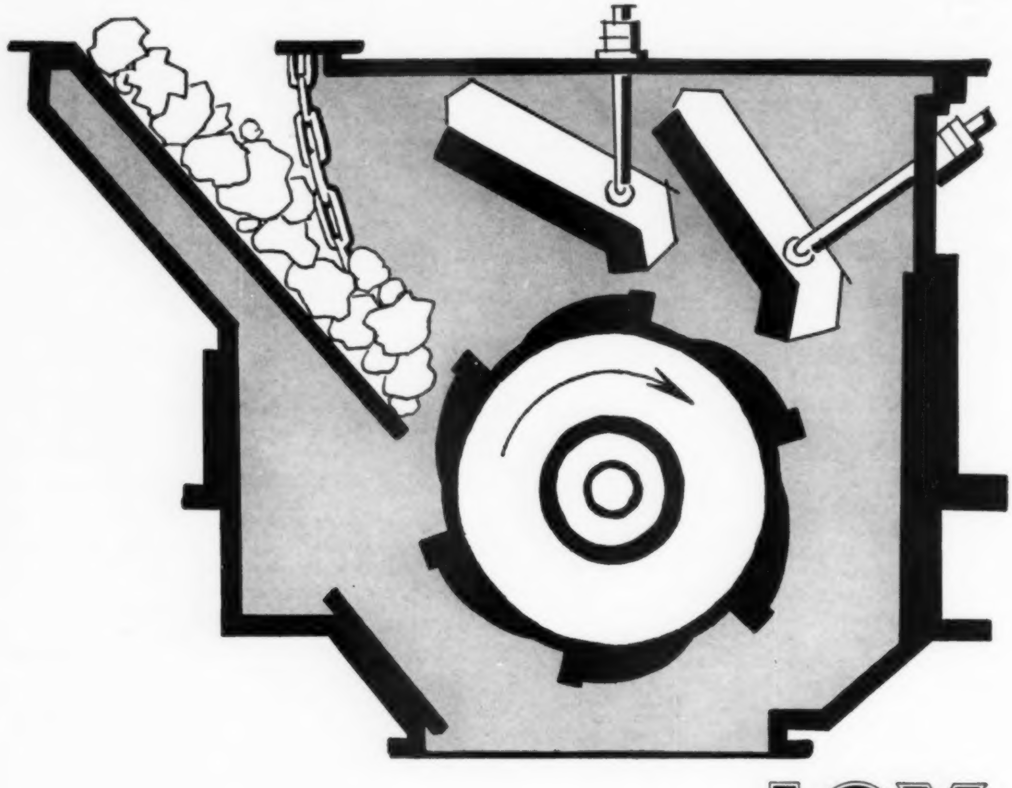
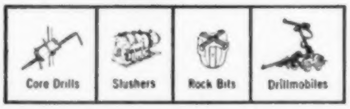


EQUIPMENT FOR MINING . . . FOR ALL INDUSTRY



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SOCIETY OF MINING ENGINEERS PRESIDENT IN 1959

J. W. Woomer

J. W. Woomer, President of the Society of Mining Engineers for 1959, is an engineer who planned his career from high school onward. His goal was his own consulting firm, J. W. Woomer & Associates, which now exists in Pittsburgh. As a sophomore he began to build experience starting as a surveyor during summer vacations. At various times in his career he has worked at all levels in the mining industry—from day laborer to mine manager. "I would recommend a similar pattern for any young man seeking a career in mining," he has declared.

Born in Philipsburg, Pa., Mr. Woomer received a B.S. from Pennsylvania State University in 1925 and an E.M. in 1931. His first full-time job was as assistant to the superintendent in the Georges Creek mining field near Frostburg, Md. He later became assistant chief engineer at the Pittsburgh Coal Co. and worked for a time for the Hanna Coal Co. When the two firms merged with Consolidation to become the largest coal mining company in the world, the new firm became his first consulting client.

Mr. Woomer has gathered wide experience all over the world. He has had mining assignments in Alaska, Argentina, Australia, Canada, Chile, China, Colombia, France, Germany, Greece, India, Manchuria, Mexico, Turkey and the United Kingdom. He has learned that many mining problems stem from public relations as well as technology, and has developed sound, practical theories from his handling of both. Although he credits John L. Lewis with the present high state of mechanization in the mining industry, Mr. Woomer strongly objects to the idea of unions for engineers. "A young man must decide whether he is going to be an educated individualist or an employee who needs the protection of a 'mother' organization."

He feels a professional attitude is essential for a mining engineer, and has approached his own work consistently with this in mind. Equal in importance to technical competence, Mr. Woomer believes, is active participation by every engineer in his professional society. He himself has long been an active member of the AIME Coal Division and in 1958 was its Chairman. He is also director of the Engineering Society of Western Pennsylvania and is a member of several other mining societies. He feels that the direct measure of a society's value is in the service it renders to its members—not just the tangible services such as publications and meetings, but also the intangible benefits that come from contact with one's fellow engineers, getting to know them and getting to be known by them.

The reorganization of AIME into the Society of Mining Engineers, the Society of Petroleum Engineers, and The Metallurgical Society, he believes, has been and will be increasingly a help to all AIME members in terms of increased services of all kinds. He assumed leadership of the Society of Mining Engineers of AIME in February with a deep sense of dedication and the resolve to bring these increasing benefits to every member of the Mining Engineering profession.

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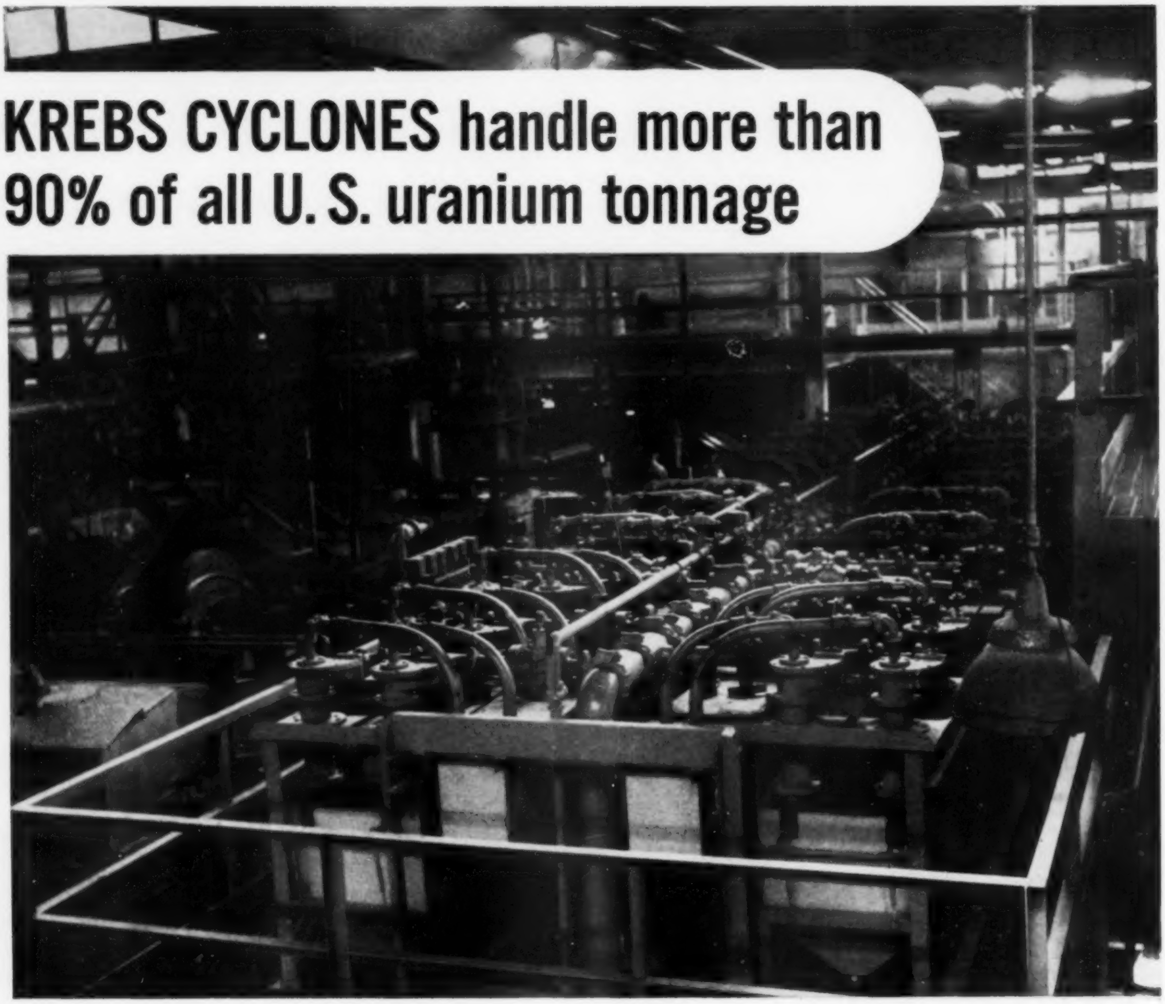

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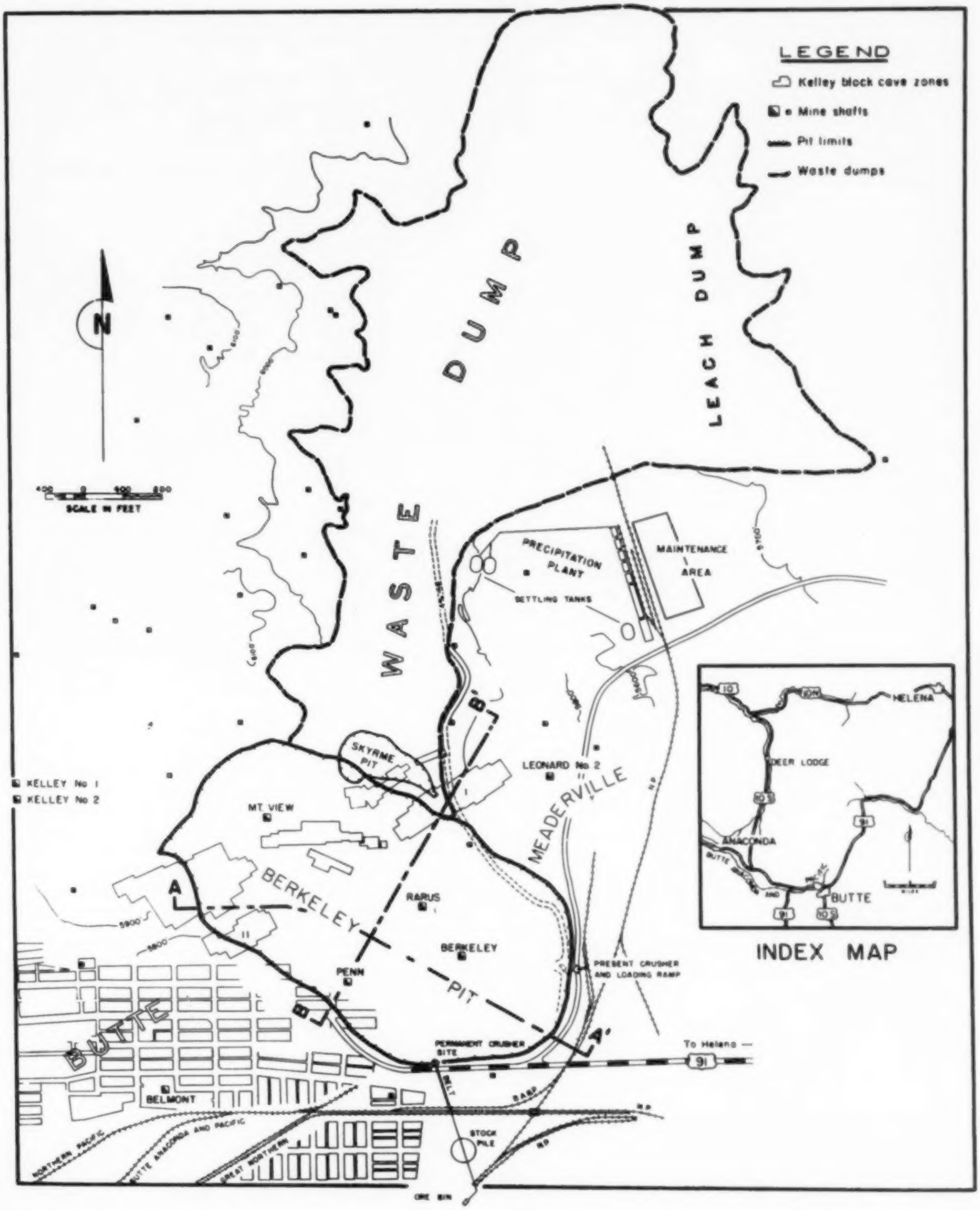
ANACONDA'S BERKELEY PIT

A four-part report on open pit mining operations



BERKELEY PIT
HISTORY AND GEOLOGY

by CHARLES C. GODDARD, JR.

Since discovery of silver-gold lode deposits in 1864, the Butte district has produced more than $2.25 billion worth of copper, zinc, manganese, silver, and gold, an unprecedented value in the mining world. Exhaustion of the gold placer deposits along Silver Bow Creek, southwest of Butte, forced the early-day placer miners to turn their attention to the bold outcrops of the many veins exposed on the hillsides above the placer diggings. These outcrops bore silver and gold, and enough so that many lode claims were located. Such famous silver mines as the Alice, Moulton, Magna Carta, Nettie, and Blue Bird were developed. These and many smaller operations contributed to making Montana the leading silver-producing state in the 1880's.

As depth was gained in the silver mines in certain parts of the district, occurrence of zinc minerals became troublesome. Zinc was not recoverable at the time, and in the chloridizing, roasting, and amalgamation of the ores, silver and gold recovery was greatly reduced. A period of active silver mining continued until 1892, when a decline in the price of silver brought the silver mining period to a close. Elsewhere in the district it was discovered that while some of the veins contained silver ores in the oxidized portions, the predominant minerals that were deeper were copper-bearing. Marcus Daly exposed rich copper ore on the 300 level of the Anaconda mine in 1891, and mining in the district disclosed that copper was destined to be developed in abundance. Early copper ore shipments were made to Swansea, Wales.

By 1900 seven or more mills and smelters were operating, and from this beginning Butte became the greatest copper-producing district in the world. By 1910 Butte had produced 6.6 billion lb of fine copper and 18 million lb of zinc; manganese mining had not been started. The numerous independent mining companies producing ores were employing many shafts; more than a few were using inadequate mining facilities. And individual operations, each on a segment of Butte's complex geologic pattern, found as depth was gained that the veins of one company entered beneath the surface of the adjoining property, giving rise to serious claims of ownership under the Apex Law. This was followed by intensive and costly litigation, finally terminated in 1910 by the Anaconda Mining Co.'s purchase of the Heinze interests.

During the second period of Butte's history, up to initiation of the Kelley mine and its successful block cave mining, the Butte Hill yielded another 6.6 billion lb of fine copper and 1 billion lb of electrolytic zinc, as well as 5 million tons of manganese oxide nodules. Eventually other metals were mined; the 78-year period ending with 1957 produced:

Copper	14,603,772,290	lb
Zinc	4,456,155,423	lb
Manganese	2,629,331,806	lb
Lead	754,799,030	lb
Silver	609,636,341	oz
Gold	2,267,213	oz

More copper was produced during World War II than during World War I, although there was continuous mining for nearly a quarter of a century between the two wars. This production record was possible only because Butte has always been able to develop one ton of new ore for each ton of ore mined, a program successfully continued at the present time. Recent work on the 4500 level, which is the deepest level in the district, has developed high grade copper oreshoots aggregating more than 2000 ft in length and containing the same type of copper mineralization that was found on higher levels. This continuing resource, accompanied by many improvements in mining methods and efficiency, has allowed Butte to continue as an important producer of mineral wealth. Much of this has been accomplished by minimizing the number of operating shafts, increasing hoisting capacity, and installing a central pumping plant. Improvements have been made in ventilation, air conditioning, and cooling of underground workings. Mine haulage and mechanization of rock handling have been modernized.

The Berkeley open pit, most recent mining unit in the Butte district, is planned to augment production of the deeper vein mines and the Kelley block cave mine. This Berkeley project, now in operation, was made possible by studies of old geologic records and incomplete sampling data on file in Anaconda's geological department in Butte. These studies indicated the existence of a supergene enriched sulfide orebody on the Butte Hill near enough to surface for possible mining by open pit. The area is highly mineralized, containing numerous veinlets and joints filled with *sooty* chalcocite coatings on pyrite and

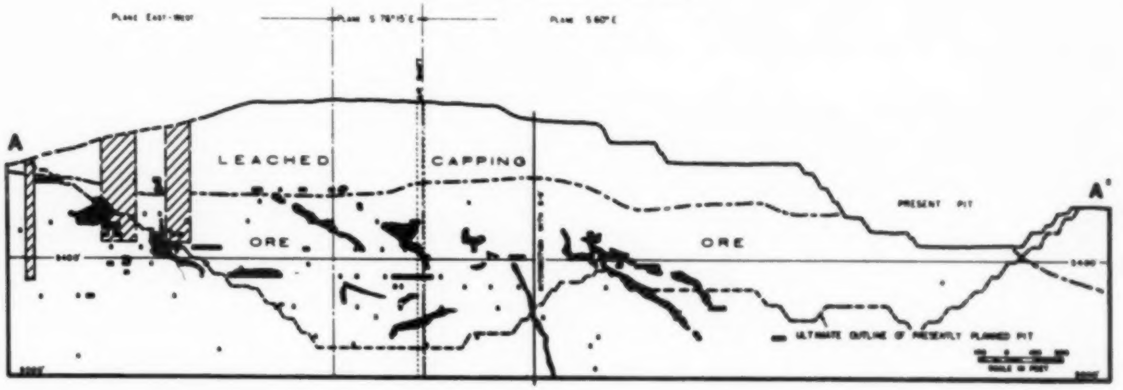
many larger veins with both primary copper minerals and added supergene chalcocite enrichment. The quartz monzonite, or Butte granite, is intensely altered, being silicified, sericitized, kaolinized, and pyritized. Much of the disseminated pyrite is coated or partially replaced by supergene chalcocite. The copper minerals other than *sooty* chalcocite are chalcocite, enargite, and minor chalcopyrite and covellite. Two quartz porphyry dikes, 50 to 150 ft wide, traverse the area. Much of the copper mineralization in the district is associated with these dikes.

Many of the larger veins in the area have been mined and filled. Hand-sorted low grade broken ore in stopes and excavated material from development headings was used for filling, and the grade of the filling is comparable to the grade of the remainder of the ore deposit. Only 5 pct of the pit area contains old mine workings.

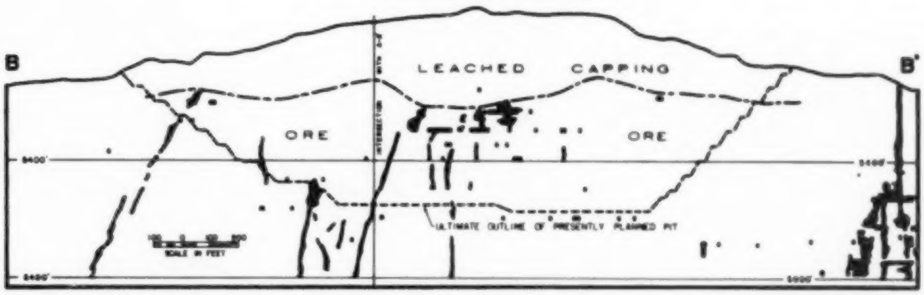
In 1952 crosscuts were driven to sample the deposit on the 200 level of the Pennsylvania mine and the 600 level of the Rarus mine. The Pennsylvania 200 crosscut is 115 ft directly above the 600 Rarus crosscut and 90 ft below the leached capping. The true grade of material in cross section is represented by shipments from these crosscuts, which were spaced to obtain an accurate sample. A total of 6138 ft of work was driven. Large-sack, hand-cut samples, mine car samples, and shipment samples were taken, and 21,179 dry tons of material were sent to the Washoe Sampler, Anaconda's custom sampling plant in Butte. This material from the development crosscuts averaged 0.83 pct Cu, 0.22 oz Ag, and 0.002 oz Au.

Because most of the upper level workings are inaccessible and crosscutting underground is slow and expensive, it was decided to use churn drill holes from surface. To determine the suitability of churn drilling for sampling this deposit, four churn drill holes were spaced over the same section sampled by the Pennsylvania 200 and Rarus 600 level crosscuts. The first three were drilled down through the crosscut to check the crosscut shipment assay data. Hole No. 1 cut ore averaging 0.57 pct Cu in a block averaging 0.59 pct Cu, according to crosscut shipment samples. Holes 2 and 3, drilled along the same lines 400 ft apart, developed a block of ore averaging 1.47 pct Cu according to drill data; the crosscut shipment samples showed the same block to average 0.80 pct Cu. Hole No. 2 cut material averaging 0.89 pct Cu, but hole No. 3 followed a small, steep-dipping veinlet and the sludge samples were not representative of the area, averaging 2.01 pct Cu for a depth of 340 ft. Four raises aggregating 210 ft driven along the holes to check the churn drill assay results demonstrated that churn drills would obtain satisfactory samples in an area of flat-dipping veins or in mineralized granite, but not where the vein structures are steep. Holes were therefore planned to miss the important steep-dipping veins, known from previous geological mapping of the underground workings; if unexpected vein material appeared in the

LONGITUDINAL SECTON A-A'



CROSS SECTION B-B'



A total of 268 holes, averaging 612.7 ft deep and aggregating 164,210 ft were drilled in the course of development. In these holes the average leached capping thickness is 275.1 ft and the ore thickness averages 175.6 ft. Solid black is old underground mining, shaded areas are Kelley block cavings.

drill cuttings, assay results were discounted before being used in ore estimates. To determine the amount of reduction required many different calculations and comparisons were made between crosscut shipment and raise shipment data and churn drill assays. The net result of these studies showed that samples of each segment of drillhole in veins assaying above 1.1 pct Cu should be reduced to 1.1 pct Cu. Original unchanged assays could be used for all other material, including country rock containing disseminated copper-bearing minerals.

Preliminary work demonstrated that sampling by churn drilling provided reliable results if the high assays were discounted in this manner. Enough new data were accumulated from crosscuts and churn drill holes to verify the information contained in the old records and to indicate an open pit operation. Churn drilling was continued to delimit the orebody. Holes were drilled on a grid spaced at 200-ft intervals, except when a hole was moved off grid to avoid drilling down a steep-dipping vein. A standard drillhole sampling procedure was adopted to meet conditions encountered in the Butte district. Since the leached capping is barren and sampling was unnecessary, general practice was to commence sampling in each hole about 25 ft above the anticipated sulfide zone. The holes were sampled for each 5-ft interval thereafter. Cuttings, sludge, and water bailed from the hole were put through a sample splitter several times. The sample was then reduced to two units. One gallon was placed in a glass container and sent to the metallurgical research department at Anaconda; the remaining half was retained in a waxed paper carton and sent to the Assay Office to determine the total copper percent, the acid-soluble copper percent, the silver, and the specific gravity of each sample.

Graphic logs on a scale of 1 in. to 10 ft were prepared by the sampler at the drill rig for each hole as the drilling progressed. Washed-panned drill cuttings were dried and cemented on the left side of the heavy paper log strip, and individual sample assays, ore averages, and waste averages were posted on the strip by the geologist as assay returns were received. All sample averages were weighted according to specific gravity, and vein material assaying more than 1.1 pct Cu was discounted to 1.1 pct. In addition to overall averages of ore and waste in each hole, averages were made for each bench interval in the proposed open pit.

The upper portion of each hole was drilled with a bit of 12-in. diam or larger to permit installing 10¾-in. casing through the leached capping and approximately 10 ft into the sulfide zone. This avoided contamination of sulfide samples by caving of the leached material above. No casing was run until the sulfide zone was reached unless caving ground was encountered. Drilling in the sulfide zone was normally done with a 9-in. diam bit, so that if caving occurred 8⅝-in. casing could be placed in the hole and drilling continued with a 7-in. diam bit. Water for drilling was lowered to the bottom of the hole with the bailer to prevent washing material from the sides of the hole and contaminating the sample.

Altogether, 268 holes were drilled, averaging 612.7 ft in depth and aggregating 164,210 ft. In these holes the average leached capping thickness is 275.1 ft; ore thickness averages 175.6 ft. Mining operations are restricted at present in the eastward extension of the orebody by surface features such as highways, railroads, and the Silver Bow Creek. West of these limitations a pit was designed by the geological and mining engineering departments. Cross sections and longitudinal sections made along the churn drillhole grid lines show the churn drill holes and assays, old mine workings, and mapped and projected geological information. These sections and the plan maps at each bench elevation were used in designing a pit to mine the greatest amount of ore with the minimum of waste removal. Pit slides were designed for a 45° slope, with 40-ft berms between benches. In areas adjacent to block cave mining the slope is planned at 38°. Benches are at 50-ft vertical intervals in the leached capping waste; originally at 25-ft intervals in the ore, these benches are now established at 33-ft intervals to provide greater efficiency in blasting and mining. All haulageways in the pit are designed for a 7° maximum grade because of adverse winter conditions.

In estimation of ore reserve, the churn drillholes were plotted on each bench plan map, together with the average assay grade for the 33 ft of hole or bench height. To determine the area to be represented by the assay average of each hole, polygons were constructed around the hole by plotting perpendicular bisectors of lines between the hole and the other adjacent holes and extending the bisector to intersect with other bisectors. Thus the area around each hole is bounded by a polygon whose sides are midway between all other holes. The area is determined with a planimeter and the volume and tons are found with the following formula:

$$\text{Volume} = \text{Area} \times \text{Height}$$

$$\text{Tons} = \frac{\text{Volume}}{11.5} \quad \text{where } 11.5 \text{ cu ft} = 1 \text{ ton.}$$

When specific gravity determinations were made from the cuttings of all holes, it was found that sulfide waste averaged 12 cu ft per ton and sulfide ore 11.5 cu ft per ton.

After it was determined that there was an orebody of enough grade and magnitude to justify open pit mining, a preliminary pit within the deposit was excavated to obtain further metallurgical, mining, and operational data. Also the shipped material was used to determine the accuracy of churn drill sample assay data. Located in an area where the overburden ranged from 125 to 230 ft thick, the preliminary pit penetrated 100 ft (four 25-ft benches) into the orebody beneath the leached capping. One of the principal reasons for the preliminary pit was to check churn drill data used to estimate the amount of waste to be removed, and the tonnage and grade of the ore to be mined. Thus, when shipments were made, the accuracy of the drill data was known. The product was concentrated separately at Anaconda to determine the treatment necessary to obtain maximum recoveries of metals.

The four benches in the preliminary pit, according to churn drill data, contained 597,966 tons of ore averaging 1.01 pct Cu, and shipments from the same benches totaled 719,051 tons of ore averaging 0.98 pct Cu—a very close check on the grade. Blasthole samples in the same area averaged 0.93 pct Cu. Results were encouraging enough to warrant proceeding with the main pit. These data may be compared with the cumulative shipments of ore at the end of June 1958, which totaled 10,047,390 tons of ore averaging 0.89 pct Cu and 0.12 oz Ag. According to blasthole samples, the average for copper was 0.84 pct Cu.

BERKELEY PIT —

MINING PLAN AND OPERATIONS

by E. O. BONNER

Open pit mining in the Butte district began with the Skyrme pit in September 1953, followed by the Berkeley and East Colusa pits in March and April 1955. The Skyrme and East Colusa pits were assigned to a local contractor, who also mined the Berkeley pit until Anaconda Co. assumed complete charge of the operation in July 1957.

The Berkeley pit was started on a trial basis. The capping of waste was stripped and enough of the orebody exposed for samples to be shipped to the Anaconda reduction department, where a beneficiation method was to be developed.

When preliminary mining and metallurgical results proved successful, long-range plans were drawn up, taking into account the following factors: pit design; necessary dump areas for leach and waste material; ore dumping site; crushing facilities; conveyance of ore to stockpile or to bins; railroad facilities to Anaconda, Mont.; capacity of the Anaconda reduction department; shovel equipment; drilling equipment; trucks, bulldozers, and graders; service facilities for mobile equipment at the pit; and change house and office buildings.

Practically all construction work involved in these plans has been completed.

The Berkeley pit is in the southwest section of Butte's mining district. Rim elevation varies from 5600 to 6100 ft above sea level; the bottom of the pit will be at 5100 ft. Named for the Berkeley mine, once a famous vein producer in this area, the pit is Anaconda's newest copper-producing unit. It adds a new mining method in the Butte district, and it supplements production from the regular vein mines and from the Kelley block cave mine.

E. O. BONNER is Assistant General Superintendent of Mines at the Berkeley Pit, The Anaconda Mining Co., Butte, Mont.

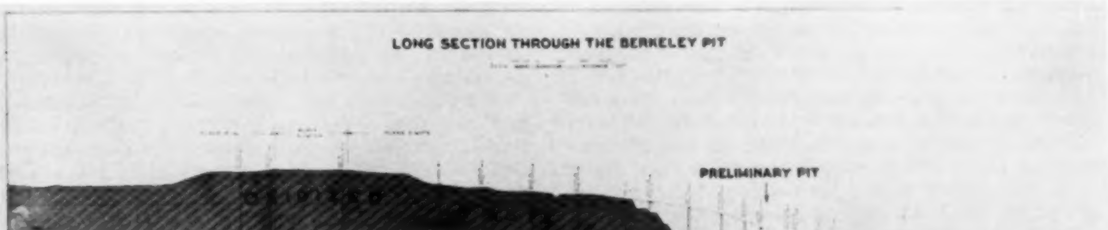
The operation was made possible by the existence of a large low grade body of sulfide ore close to surface, by application of the open pit method with large power shovels and trucks, and by adequate dump areas for leach and waste material.

The existence of such an orebody was brought to light by detailed structural and mineralogical data accumulated for many years by the geological department, by development work done in the Kelley mine, and by certain crosscuts and churn drill holes. To delimit the orebody, churn drill holes were spaced at 200-ft intervals.

Standard procedure was to sample each of these holes at 5-ft intervals, starting about 25 ft above the sulfide zone.

The finished pit perimeter will measure 4600 x 2600 ft at the top, when stripping of waste is completed.

Two hundred sixty-eight holes were drilled for a total footage of 164,210 ft, an average of 612.7 ft per



This section shows the orebody which still has 123 million tons of reserves as of Feb. 1, 1959.

Summary of Berkeley Pit Costs, Six-Month Average

	Cost Per Ton of Material
Blasting:	
Blasthole drilling	0.0032
Blasthole drilling power	0.0004
Blasting	0.0131
Repairs to drill equipment	0.0010
Total Blasting	0.0177
Shovel Loading:	
Shovel loading	0.0141
Shovel loading power	0.0012
Repairs to shovels	0.0087
Total Shovel Loading	0.0240
Truck Haulage:	
Truck haulage	0.0558
Road maintenance	0.0021
Dump maintenance	0.0028
Repairs to trucks	0.0285
Repairs to tractors	0.0067
Repairs to graders	0.0007
Repairs to service trucks	0.0014
Total Truck Haulage	0.0980
Crushing, Conveying, and Loading Ore:	
Crushing, conveying, and loading ore	0.0080
Crushing, conveying, and loading ore power	0.0012
Repairs to diesel locomotives	0.0001
Repairs to crushing plant, conveyor, and bins	0.0030
Total: Crushing, Conveying, and Loading Ore	0.0123
Miscellaneous Expenses:	
Superintendence, technical, and clerical	0.0112
Service and other miscellaneous	0.0148
Service and other miscellaneous power	0.0003
Fire expense	0.0001
Move power lines	0.0004
Maintain airways	0.0007
Maintain airways power	—
Move miscellaneous equipment	0.0001
Holiday pay	0.0017
Workmen's compensation	0.0020
Vacation allowance	0.0017
Unemployment tax	0.0009
Federal insurance contribution	0.0020
Total, Miscellaneous	0.0359
Total	0.1879
General expense	0.0406
Total	0.2285
Amortization of Development:	0.0616
To combine development and mining costs	0.0657
Total: All Expenses	0.2244

hole. Average leached capping of waste is 275.1 ft and ore thickness 175.6 ft, as determined from the churn drilling program.

To clarify pit design, cross sections and longitudinal sections were made up through the churn drill hole grid lines. In addition, plan maps of each proposed bench were made up. The pit was designed according to these sections and plan maps, using a 45° slope for the sides. Bench heights were 25 and 50 ft, but eventually a 33-ft bench proved most economical in holding waste removal to a minimum.

A valley northeast of the operation area meets the disposal requirements for leach and waste material to completion of the pit.

To facilitate working equipment efficiently under the worst possible weather conditions, all haulageways in the pit area are laid out on a maximum grade of 7 pct. Grades on haulage roads outside the pit proper, to waste and leach dump areas, are set at 4 pct grade. Very close engineering control is required to develop the pit according to plan, as well as to keep production at desired grade and tonnage.

The Berkeley pit has been in operation since March 1955, at which time a waste stripping program was inaugurated. The first ore was shipped Dec. 19, 1955. Through Feb. 1, 1959, 14,617,000 tons of ore have been shipped; a total of 2,675,000 tons of leach material stockpiled; and 49,842,000 tons of waste removed.

As of Feb. 1, 1959, an ore tonnage of 123 million remained in reserve. To mine this ore, 138 million tons of waste and 40 million tons of leach material will be removed at a stripping ratio of 1.47 to 1. Ore grade averages 0.75 pct Cu and 0.17 oz Ag; grade of the leach material averages 0.2 pct Cu.

Past production has been as follows: by the end of 1955, 3000 tpd; by the end of 1956, 10,000 tpd; by the end of 1957, 17,500 tpd. As of February 1959, production was 28,500 tpd.

Pit Estimates: Estimates of ore production and removal of leach and waste material are made on a monthly, bimonthly, semiannual, and yearly basis. Such estimates are necessary in overall planning of the pit for future ore production, removal of leach and waste material, and equipment requirements.

Measuring Methods: Base values in the pit are surveyed in by triangulation from a permanent base line. According to these values, stations are set for surveying toes and crests of each bench with a plane table and alidade.

Outlines of benches (toe, crest, and median line) as they appear on the first of the month are transferred to permanent record volume sheets. The surface area between median lines is planimetered, and the resulting area is multiplied by the bench height and divided by 27 to arrive at the yards removed from any particular bench. Total yardage of all material removed from the pit is thus computed and converted to tons.

The factors for computing tons are 12 cu ft per ton for oxide material and 11.5 cu ft for sulfide material.

On benches that contain ore, leach, and waste material, the ore and leach splits are arrived at from the truck tallies in tons. Yardage of waste is calculated by subtracting the yardage of ore and leach (converted to yards from tons) from the total yards on the bench.

Holdbacks from benches are calculated and recorded and, when removed from the pit, are credited to their respective bench.

Crushing Plant, Conveyor, and Storage System: The Berkeley pit crushing plant and conveyor system are capable of moving 2000 tph from the truck dumping site to the stockpile area or to the ore bins. The stockpile area has a capacity of 31,000 tons, the ore bins 5000 tons.

MINING OPERATIONS IN THE BERKELEY PIT

Drilling: Drilling equipment for the pit consists of three track-mounted electric rotary drills and one truck-mounted diesel-powered rotary drill. In the drilling operation tricone bits are used exclusively, a 9-in. size with the electric rotary machine and a 6¼-in. size with the diesel machine.

After a number of different makes of tricone bits had been evaluated under various drilling conditions, operations were standardized to the use of two types—a medium formation type of bit and a hard formation type. Footage per bit varies between 3500 and 5000 ft ore and bit costs from $0.035 to $0.09 per ft.

In an 8-hr shift the diesel-powered drill averages 604 ft and the electric drill 521 ft. The six-month average cost per foot of hole drilled is $0.2414.

Distance between drillholes for different rock types is approaching optimum spacing. The present 30-ft spacing between holes in the ore and the 25-ft spacing in the waste have given good fragmentation. Possibly the hole spacing in the ore may be ex-

To be able to work pit equipment efficiently under the worst possible weather conditions all haulageways in the pit area are a maximum 7 pct grade. Roads outside the pit are 4 pct maximum grade.

tended beyond 30 ft in some areas. Distance from toe varies from 21 to 33 ft. Depth of blastholes is 38 to 40 ft, or 5 to 7 ft below bench grade.

A hard and fast rule for hole spacing is not practical for all areas in the pit. By correlating drill speeds with hardness of ground, patterns of hole spacing (down to 18 ft between holes) have been laid out for hard areas in the pit. This operating policy has resulted in good fragmentation of the broken rock in hard areas and has eliminated delays caused by hard digging, rough bottom, redrilling, and reblasting.

Standardized spacing for soft, medium, and hard ground, as correlated with drill speed for the respective type of ground, have reduced drilling costs materially. Present blasthole drilling cost per ton of material is $0.0177.

Sampling: Most samples taken from the pit are drill cuttings from the blastholes. Each drill is equipped with a baffle placed around the drill steel; this baffle is 36 in. above the ground and has a skirt of fan bag material extending to the ground. Cuttings are blown from the hole by compressed air to surface through the space between the wall of the hole and the drill steel. At surface these cuttings are stopped by the baffle, drop to the ground within the skirt area, and form a cone-shaped pile around the collar of the hole. After a hole is collared, a pan 17 in. high, 28 in. long, and 2 in. ID is placed at right angles to the drill steel, the 28-in. length forming the radius of the circular pile. The cuttings that drop into the pan make up the sample of material through which the drill bit has passed. Samples weigh 25 to 30 lb. Since blastholes are drilled below bench level, the sample pan is removed as soon as bench elevation is reached so that it will contain only material to be shipped from a blast. Each blasthole sampled is numbered, surveyed, and located on a map with its copper value. All samples within a given area are weighed and averaged to determine the assay value of any day's shipment. Shipments are maintained at the average pit grade, which is made up by blending low grade and high grade material running from 0.5 pct to more than 1 pct Cu. Blasthole samples have proved accurate to within 0.05 pct Cu when compared to actual smelter returns. They are excellent guides for controlling grade of daily shipments.

When it is necessary to ascertain tonnage and grade of material that may have to be segregated from the ore stream and placed on separate stockpiles for future treatment, cut sampling is done across gobs and adjacent mineralized granite, which contain major quantities of copper sulfate and other post-mine minerals detrimental to smelter recoveries.

Other samples representing all types of ore, gob, mineralized granite, and quartz porphyry are taken by approved methods and sent monthly to the Butte and Anaconda research laboratories in a continuing attempt to increase copper recovery from the ore.

Geologic Mapping: To provide data for pit expansion, geologic mapping of all benches has been carried out since the operation was started. Notes on

structure and alteration that are taken on any bench are aids in forecasting results for benches at lower horizons. They also guide the determination of bench slopes in any pit areas that might be subject to rock slides.

Blasting: Blasting is done with ammonium nitrate prills of fertilizer grade, mixed with diesel fuel oil in the ratio of three quarts of oil to an 80-lb bag of ammonium nitrate. Present practice is to unload and store the bags of prilled ammonium nitrate on active benches in approved locations and soak them with diesel fuel oil for at least 72 hr so that capillary action will coat the prills thoroughly with oil. These prepared bags of ammonium nitrate prills are distributed by truck to the collars of holes to be blasted. Two hundred to four hundred pounds of the prills are used per hole in the waste and 320 lb in the ore.

The following primers have been successful in the detonating operation: 5½-lb cans of Nitramite; 3-oz Hercules XC 49 pucks; 10-lb sticks of 60 pct dynamite; and 3 and 5-lb bags of Nitramite. At present nearly all detonation is accomplished with the 3 or the 5-lb bags of Nitramite. Since there are homes in the vicinity, explosives have been limited to 3200 lb per shot. The millisecond firing decreases vibrations, lessening the possibility of damage to residences.

Loading and blasting is handled by a four-man powder crew, which includes a truck driver. After the correct number of bags have been placed around each hole to be loaded, half of the powder is poured into the hole. A powder man attaches the primer to the end of a roll of Primacord and lowers it, cutting the Primacord about 2 ft above the collar of the hole. The remaining powder is poured into the hole, which is then filled with drill cuttings (stemming). After all holes for a blast have been loaded, the Primacord truck line is strung and the loose ends of the Primacord projecting from each hole are tied to it. Then No. 9 and No. 17 millisecond connectors are placed in the main trunk line between each hole and a Primacord pigtail with an electric blasting cap is attached to the firing end of the trunk line. Electric blasting wires are strung from the pigtail to a battery-operated blasting box. During the last part of this hook-up operation, the blasting area is cleared, and only one man of the powder crew is active in the blasting area. The blast is set off by an electric push-button blasting box located a safe distance away.

Breakage and fragmentation from ammonium nitrate blasting has been very good. There has been some secondary blasting in the very hard rock areas, but generally this is not required. The six-month average cost to break one ton of material is $0.0177. The material broken varies from 4.22 to 6.23 tons per lb of powder, averaging 5.35 tons.

Shovel Loading: Five 6-cu yd electric shovels and one 3-cu yd diesel-powered shovel are worked in the pit. Practically all loading is accomplished by the electric machines; the diesel is used mainly for cleanup and for loading in areas where the electric type cannot be used. To work the electric shovels

Left: Blasting is with fertilizer grade ammonium nitrate prills mixed with diesel fuel oil; the present primer is Nitramite. Breakage and fragmentation with ammonium nitrate has been good.

to best advantage, a bench width of at least 100 ft is required. No pushbacks in the Berkeley pit are less than this figure. By maintaining at least this width, it is possible to load into trucks on both sides of the shovel, which makes for an efficient shovel operation. Bulldozers keep the truck loading positions at shovel locations leveled off and cleared of boulders.

All shovels are averaging 6710 tons per 8-hr shift, and at certain times shovels in waste have attained more than 10,000 tons per shift. Average shovel tonnage includes time spent changing crowd cables, haul-back cables, hoist cables, power failures, and other miscellaneous delays. An average of 2 hr is required to change each of the three types of cables.

Automatic oilers have been installed on all the electric-powered units. This has greatly increased the tons per cable. Average for a hoist cable is 950,000 tons; for a haul-back cable, 925,000 tons; and for a crowd cable, 300,000 tons.

Cost of loading, six-month average, is \$0.024 per ton of material moved. Cost of repairs to shovels, six-month average, is \$0.0087 per ton of material moved.

Haulage: Diesel-powered trucks remove broken material from the pit. Each truck trip averages 34 tons, totaling some 1008 tons per 8-hr shift. Haulage distances vary from 3000 to 13,000 ft. The six-month trucking cost is \$0.098 per ton of material, including a tire cost of \$0.021.

Trucks are available about 84 pct of the time. Out-time includes fueling and scheduled maintenance checks. Cost of repairs to trucks is \$0.0285 per ton of material removed.

Surfacing of Permanent Roads: Permanent roads within the pit area and to waste and leach areas are surfaced with 18 in. of gravel aggregate, compacted by a pneumatic tire roller.

Besides surfacing permanent roads with the compacted gravel aggregate, another method of surfacing, which has been successful at Anaconda's Yerington property, will be tried out in the near future: 2 in. of roadway surface will be scarified, windrowed and blade-mixed to uniform consistency with a 10 pct lignite sulfanate solution at 2 gal per square yard of roadway. The resulting material will be relaid and then rolled with a pneumatic roller. Truck traffic will complete hardening of the roadway, which will be sealed with a high viscosity emulsified asphalt at 0.4 gal per sq yd. Then it will be cured for 24 hr and sanded with 15 lb of sand per sq yd.

Road and Dump Maintenance: Bulldozers are used at leach and waste unloading sites to keep those areas leveled off and cleared of boulders. Road graders constantly patrol the haul roads to the ore, leach, and waste unloading sites on all shifts to maintain roads in good condition and cleared of boulders and rocks at all times. This cleanup operating, using bulldozers and road graders, is absolutely necessary to reduce excessive tire damage.

BERKELEY PIT—
CRUSHING AND CONVEYING SYSTEM

by F. RALPH

Berkeley's crushing and conveying system is best explained by the accompanying sketch. At the extreme left of the diagram, which represents the north end of the present rock-handling system, are the truck-dumping station and crushing plant, adjacent to the pit. At the extreme right, or southern end of the project, is the site chosen for storage bins, since railroad facilities near this vicinity could be made readily available for Berkeley's operation.

The crushing and conveying system is a series of installations consisting of truck-dumping facilities, a screening and crushing plant, five main conveyor belts, stockpiling equipment, and storage and railroad car-loading facilities capable of handling 2000 tph of ore.

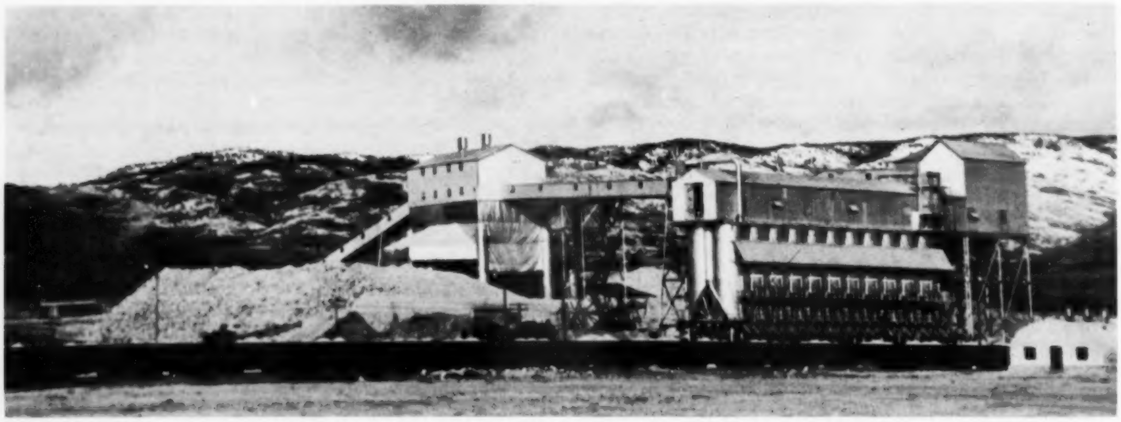
Between the pit on the north and the railroad facilities on the south the area was transversed by a U.S. highway; a city street; the Northern Pacific and the Great Northern railroads; a Butte, Anaconda & Pacific spur to one of the Anaconda operations; and any number of telephone and power lines. With such definite boundaries at each end and so many obstructions between, it was decided to go underground with part of the project.

Because of the extensive surface installations and obstructions a 700-ft tunnel was driven on 12 pct incline, terminating 160 ft below the crusher site.

TRANSFER TUNNEL

At a point south of the Great Northern Railroad a tunnel was driven approximately 700 ft northerly on a 12° decline terminating 160 ft directly under the crusher site. For most of its length this tunnel is 8 ft 6 in. by 13 ft in semicircular cross section. A 100-ft length at the northern end of the tunnel was enlarged to 18x40 ft to accommodate the mechanical and electrical installations necessary at this point of transfer in the system.

The tunnel was driven by conventional drilling, blasting, and mucking methods. The muck, however, was transported up the 12° incline by 6-yd Dumptor equipped with scrubbers to clean the exhaust. In sections of the tunnel where heavy ground was encountered steel sets and landing mats were used to facilitate mining and hold the ground until concreting could be completed. Except for small sections



Two rows of circular steel-bottom ore bins provide 5000 tons of storage at rail car loading point.

where it was possible to pour concrete through boreholes from the surface, most of the concrete was pumped into forms by a pumpcrete machine located at the tunnel portal. Four-yard transit truck mixers delivered concrete to this machine from the central concrete plant.

The tunnel was terminated at this specified horizon below the crushing plant to provide the difference in elevation necessary to satisfy plant design. The top 60-ft depth was required to accommodate the truck hoppers and feeders, the vibrating screen, and the crusher proper. The connecting 60 ft between these two excavations is a rail-lined raise 16 ft in diam that serves the dual purpose of storage and surge bin.

South of the Great Northern Railroad, except for that section under the stockpile, the conveyor system is above-ground and is housed in a structural steel gallery that is supported by five steel towers varying in height from 60 to 90 ft. This gallery terminates at the railroad storage bins.

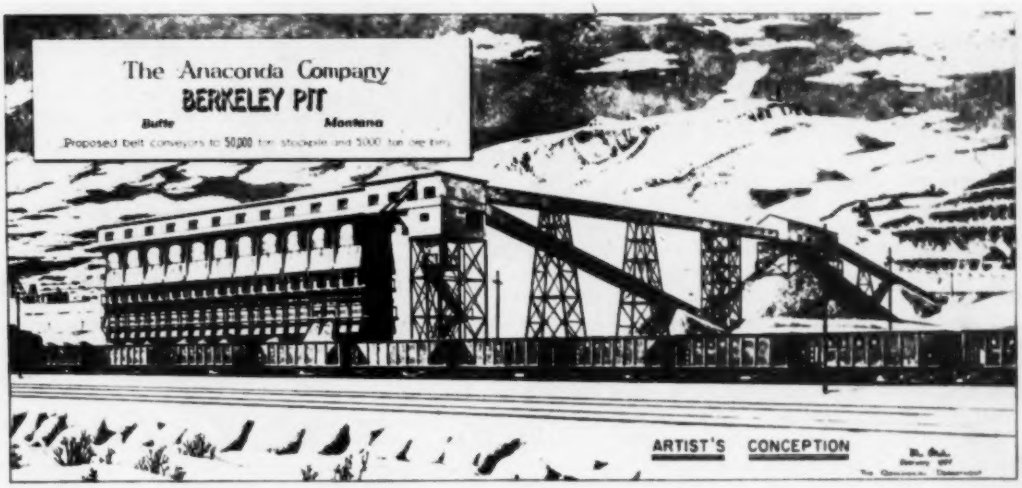
It is a matter of pride that even though the sections of conveyor galleries varied in length from 80 to 106 ft and weighed up to 20 tons, they were completely assembled on the ground and raised into position without the use of falsework, staging, or guy lines.

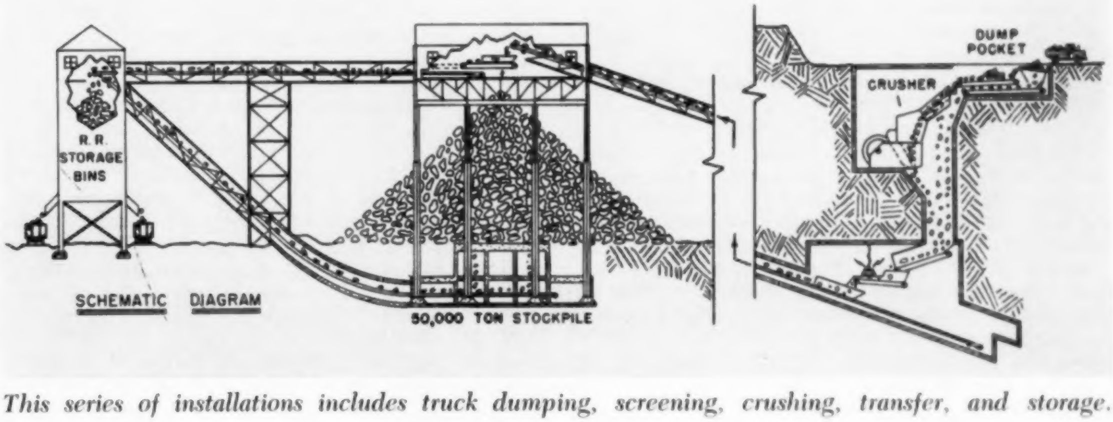
STORAGE FACILITIES

At the railroad car loading site two rows of circular steel-bottom ore bins provide 5000 tons of storage. The 75-ton gondola-type ore cars are loaded by means of 44 pneumatically operated gates.

In erecting this type of bin structure at the Kelley mine it had been found very difficult to maintain alignment of a series of circular steel plates for welding. To defeat this problem and eliminate the need for elaborate scaffolding, a welding jig was rigged on the ground to weld the plates to a template. When the structural skeleton of the bin was completed, the preassembled sides were raised into position, thus reducing to a minimum the amount of welding necessary in place.

Additional storage is provided in a stockpile area north of the railroad loading bins. This stockpile is placed over a reinforced concrete underground structure 60x70x35 ft high that is capable of supporting the 35,000 tons of ore sometimes stored here. Housed in the four corners of this structure are four 5x24-ft pan feeders equipped with 60-in. dribble belts that draw down the stockpile as desired, loading it onto the No. 3 conveyor belt to be transported to the 5000-ton bins for loading.





This series of installations includes truck dumping, screening, crushing, transfer, and storage.

CRUSHING PLANT

At the crusher itself, the feeders, vibrating screen, and crusher foundation settings vary in depths from 20 to 60 ft below the surface, which at this location is the 5600 bench of the pit. It was originally planned to excavate this 60 ft by open cutting, maintaining the walls of the cut as close to vertical as possible. This idea was abandoned at a depth of about 24 ft because of heavy side walls and the possibility of excessive overbreak. At this point a method of excavating and concreting from the surface down was inaugurated. This was accomplished by first pouring a concrete bearing ring 6 ft thick and 10 ft wide around the excavation 24 ft below the surface. The top sections of walls were then poured on the ring, using both inside and outside forms. From this bearing ring down, the excavation was very closely controlled. Overbreak was held to a minimum, only one form was used, and concrete was poured to ground as the work progressed. When the concrete structure and equipment foundations were completed, the steel structure that houses and supports a 50-ton bridge crane was erected. This crane was installed to facilitate construction and maintenance of the crusher. The 60x84-in. Blake-type jaw crusher, weighing about 255 tons, is capable of crushing 800 tph of ore at an 8-in. setting. The heaviest pieces handled in its construction were the stationary and swing jaws, each weighing 45 tons.

ROCK FLOW

Truck to Railroad Car: Trucks may dump simultaneously at any one of three stations. At the east and west stations, which are for run-of-mine rocks, ore is dumped on the main 5x30-ft pan feeder and fed over the 6x14 vibrating screen that segregates the −8-in. material and propels the larger material into the crusher. The north dumping station is for gob rock that contains timber from old workings. This material is deposited onto a 13-bar wobbler feeder 5 ft wide which screens out the −8-in. material and forces the oversize rock and timber toward the pan feeder. At point of discharge from the wobbler the timber is picked from the ore stream and the oversize rock continues on with the run-of-mine rock to the crusher.

The −8-in. material segregated by the wobbler is deposited on a 60-in. conveyor belt and discharged into the surge bin, which also receives the undersize from the vibrating screen and the oversize that goes through the crusher and is reduced to −9 in. To insure cheaper maintenance and longer belt life, all material reaching the conveying system is −9-in.,* thus insuring cheaper maintenance. At the bottom of the surge bin is another 5x30-ft pan feeder that sends the ore over a set of grizzlies and onto a 6x30-ft flat picking belt, where undesirable foreign material and tramp iron are extracted from the flow. Tramp iron is spotted by an electronic detector connected in series with the picking belt control and is picked out by a 60-in. electromagnet. Any magnetic material passing through the field of the detector actuates a relay that shuts down the picking belt. Then by jogging the belt motor with a manual override control the tramp iron can be moved toward the head pulley and extracted from the bed of ore.

* It might be well to mention at this point that all of the main belts in this system were spliced by vulcanizing in accordance with the manufacturer's specifications.

From the picking belt ore is discharged over another set of grizzlies to the No. 1 belt. This 6-ply belt, the largest in the system, is 2126 ft long and 48 in. wide. It is designed to operate at 200 lb per inch width per ply at a speed of 400 fpm on a 12° incline. The belt has 5/16-in. standard commercial top-quality rubber cover, including rayon transcord breaker and 3/32-in. back cover. Piles are woven in the lengthwise direction with special rayon fiber of the highest tensile strength commercially available. Strong flexible nylon fiber is used in the crosswise strands.

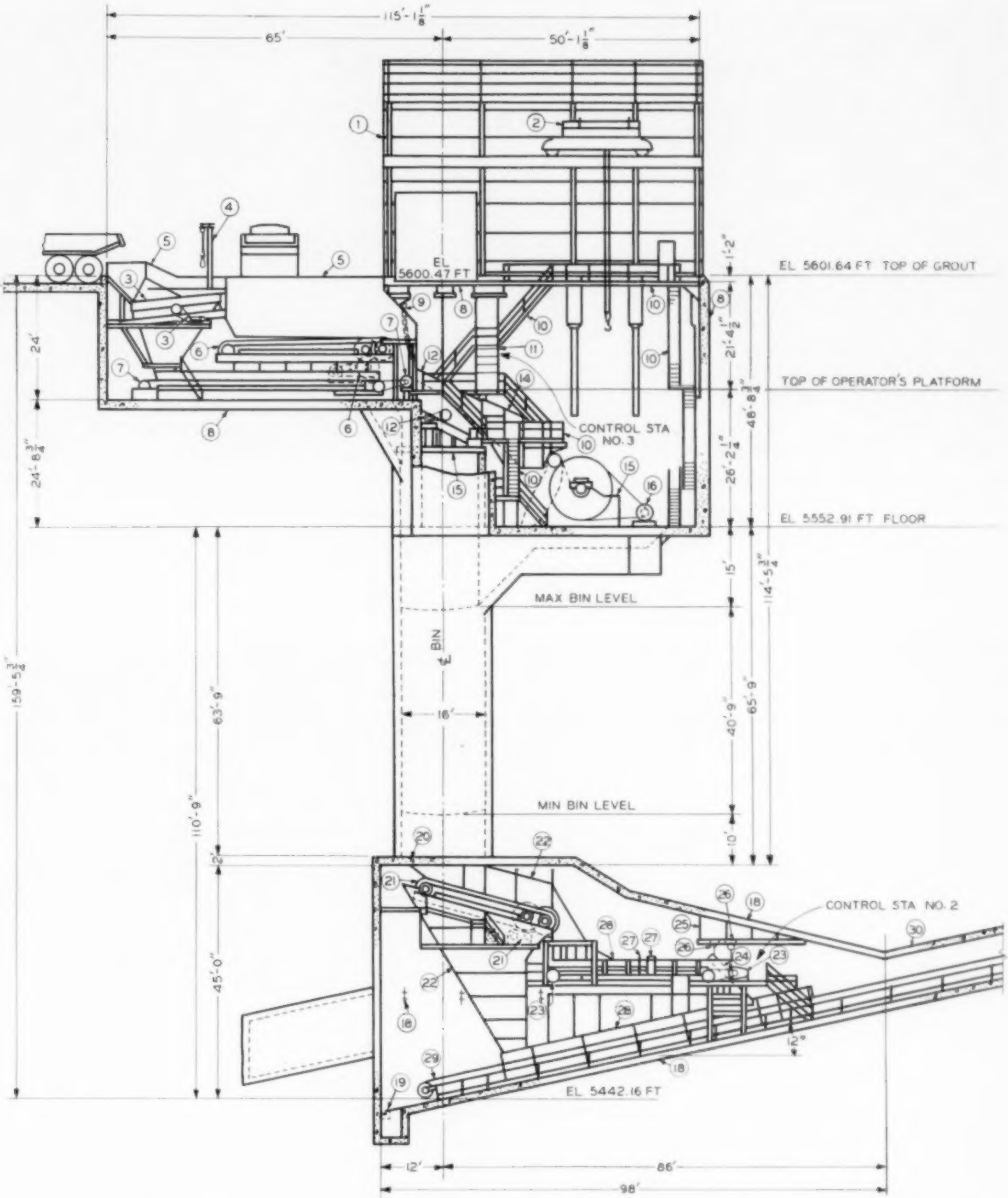
This conveyor, 1018 ft from head to tail pulley, is powered by a double 30-hp tandem drive. The belt discharges either onto the No. 5 shuttle belt and on through the system or into the stockpile. The No. 5 shuttle belt, 60 in. by 20 ft, is mounted on a movable structure that can be shuttled in or out of rock-carrying position to suit the operation. When working, the No. 5 belt discharges the rock onto the No. 2 conveyor belt.

The No. 2 belt is 5-ply and 48 in. and about 1045 ft long. It operates at 400 fpm on a decline of 2° 12′. This conveyor, which is 504 ft from head to tail pulley, is powered by a 30-hp motor. Ore from the No. 2 belt is discharged onto the No. 4 conveyor.

The No. 3 conveyor is a 5-ply, 48-in. belt operating at 400 fpm on a 12° incline from under the stockpile. It is approximately 1155 ft long. This conveyor, 558 ft from head to tail pulley, is powered by a single 300-hp motor. Ore from the No. 3 conveyor belt discharges onto the No. 4 conveyor, a 6-ply, 60-in. belt. With the aid of a tripper, the No. 4 conveyor discharges the ore received from both the No. 2 and 3 belts into the 5000-ton railroad ore bins. The No. 4 belt is 195 ft from head to tail pulley and is 460 ft altogether. This belt travels at 500 fpm and is capable of handling 400 tph, which is the combined capacity of both the No. 2 and 3 conveyors. The tripper that works in conjunction with No. 4 conveyor is mounted on a carriage that travels back and forth over the length of the bin, controlled as to dumping position either manually or automatically depending on whether the ore is to be spotted in one place or spread the length of the bin.

Control: The entire system is operated from three independent control stations. Station No. 1 is located in the transfer tower at the junction of the No. 2, 3, and 4 conveyors. Station No. 2 is located at the picking belt near the loading point of No. 1 conveyor and No. 3 station is located at the crusher. The control panel at station No. 3 controls the crusher, the vibrating grizzley, the wobbler belt, and the 5x30 pan feeder. The wobbler is controlled manually to allow the many stops and starts necessary to pick the timber from the ore stream. All units of this system are interlocked through the control circuits so that no unit can be started out of sequence. Otherwise a great deal of trouble could result from starting the wrong piece of equipment at an improper time. This control feature is maintained throughout the plant.

The control panel at Station No. 2 controls the 5x30 pan feeder at the bottom of the surge bin, the picking belt, and the No. 1 conveyor. The operators at the No. 2 and 3 control stations are in constant communications through the interplant phone system. By controlling speed rates of the 5x30 pan feeders they can iron out the surges of irregular dumping from the trucks to a smooth and constant flow throughout the balance of the system.



This elevation shows equipment at control stations No. 3 (above) and No. 2 (below). Conveyor No. 1 carries material to right from control station No. 2 to a shuttle belt over the 35,000-ton stockpile. Conveyor No. 3, located in a tunnel beneath the stockpile, carries material to a transfer tower from which the 5,000-ton railroad ore bins are fed.

The panel at Station No. 1 controls the No. 5 shuttle belt, No. 2 conveyor, No. 4 conveyor and tripper, the four stockpile feeders, and the No. 3 conveyor. By shuttling No. 5 conveyor in or out of position the operator at this station controls the disposition of the rock at all times, either to the stockpile or to the bins to be loaded into cars for transportation to Anaconda.

Summary: Installation of the crushing and conveying system at the Berkeley required 56,000 cu yd of plant excavation, 900 cu yd of concrete and 520 tons of reinforcing steel, approximately 2050 tons of structure and equipment, electric motors with total installed capacity of 2250 hp, about 4½ miles of conduit, 18 miles of power feed and control wire, and 5600 ft of conveyor belting capable of handling 2000 tons of crushed ore per hr.

Service area includes: a) combination washrack and paint shop; b) main shop building; c) heavy equipment warehouse; d) pit office and change house; and e) electric heater island.

BERKELEY PIT — SERVICING MOBILE EQUIPMENT

by P. M. YOUNG

An important part of the planning for the Berkeley pit operation included complete facilities for the repair, servicing, and maintenance of all mobile pit equipment. In order to provide adequate facilities for these operations the following factors had to be considered:

- Number, size, and classes of equipment.
- Schedule of pit operations.
- Location of plant.
- Sufficient area to locate all facilities needed.

The schedule of pit operations varies and all possible combinations of operations had to be considered. The service plant is easily accessible to the pit operations from its location on a large flat area a short distance from the east rim of the pit and waste dump. A short access road extends from the north end of the plant area up to the waste haul roads, where access to any part of the pit is convenient. A short branch road to the south connects with the main highway and on the west boundary of the area is a spur of the Northern Pacific Ry.

In this service area are located the pit offices and change house, main shop building with main service bays, various sub-repair rooms, shop office, warehouse office, tool room, parts room, and sanitary facilities. There is in addition a warehouse for heavy equipment parts, a combination wash rack and paint shop, fuel storage tanks, pump house, fuel dispensing pumps, and an electric heater island for plugging in electric heaters on parked trucks in cold weather.

The entire service section is located on a well-drained area, about 720 ft wide by 1000 ft long, enclosed by an 8-ft Cyclone fence. The area is also well lighted for night operations.

The lower floor of the change house building houses the pit superintendent, pit foreman, pit shift bosses, powder foreman, engineers, timekeeper, and superintendent's clerks. On the upper floor is a modern change room, well lighted and heated and equipped with steel wire baskets for 400 men. Shower and toilet facilities are included.

The main shop building is a steel framed structure covered on the outside with corrugated sheet steel, insulated with 1½-in. Fiberglas and covered on the inside with corrugated galvanized steel siding. This building is 90x360 ft and is divided along the long axis into two parts. The main truck bay area is 50x340 ft and the sub-shop area is 40x300 ft. The main truck bays are serviced by a 20-ton pendant-controlled crane. Main columns of the building are on 20-ft centers so that each bay has ample room for repairing a 50-ton truck.

The first two bays on the south end of the building are reserved for tractor, grader, and drill rig repairs. The third bay is used for tire repairs and service. The next 12 bays are for truck repairs and the next two for lubrication. The four grease pits in these two bays make it possible to service four trucks at a time. Since the pits run the full building width, trucks can be driven straight through.

The extreme north end bay is used for the electric shop, toilets, lunch room, and oil house. The upper floor of this bay provides offce room for clerks and garage superintendent. The first four bays in the sub-shop area at the south end are used for the plate

and welding shop. The next three bays are for transmission, differential, and brake repairs; the next two house the tool room and spare parts; the following two bays are for converter, fuel pumps, and injector repairs; and the last four are for engine overhaul and testing.

The building is well-lighted, heated and ventilated. Ventilation includes an exhaust gas collecting system for use when engines must be operated in the building. This same system connects the grease pits to protect against any accumulations of toxic gas.

Along the walls of the building are workbenches equipped with vises, parts cleaning equipment, water and air hoses on reels. Handy electrical convenience outlets and plug-in sockets for electric welders are also spaced at intervals. Floor stand grinders are provided where necessary. Electric operated roll-up doors are provided for all bays.

Along the east side of the building, running full length, is a 40-ft-wide concrete apron. A large area surrounding the building is used as parking space for equipment awaiting repairs or completed.

Weight of the equipment called for special consideration of the type of surface to be used. Compacted crushed stone 24 in. deep was decided on. The area was first excavated to depth and ¾-in. stone place in 6-in. layers and compacted by wetting and rolling. This surface has given good service.

Sub-Repair Shops: The list on the following page shows principal tools and equipment.

Adequate spare parts are carried in stock for all the equipment, except for parts that are available from local stores. An efficient Kardex system is used to check on 4800 stock items.

Oil and grease drums are stored in the oil house and are piped to the dispensing hoses at the grease pits. Air operated barrel pumps supply the power to pump the oil and grease.

Outside the north wall of the main shop, in an underground concrete room, are two 10,000-gal motor oil tanks for the No. 20 and No. 30 motor oil. Air-operated pumps deliver oil to the grease pits.

Diesel fuel is stored above ground in four 20,000-gal tanks. Three of these are for No. 2 diesel fuel and one for No. 1 diesel fuel. The No. 1 fuel is used for special equipment and as an additive to ammonium nitrate for blasting in the pit.

Gasoline is stored in two 10,000-gal underground tanks to the north of the diesel fuel tanks. Each gasoline storage tank has a vertical turbine pump mounted on the tank to pump the fuel to the dispensing stations.

A concrete pump house is located next to the diesel storage tanks along the railroad spur. The pump house has four motor-operated pumps and two fuel oil filters and dehydrators. All of the fuel

The first two bays at the south end of the building are reserved for tractor, grader, and drill rig repairs. The third bay is for tire repairs. The next 14 bays are for truck repair, and for lubrication.

PIT EQUIPMENT REQUIRING SERVICE

Forty-three 34-ton rear-end dump trucks
Five 6-cu yd electric shovels
One 3-cu yd electric shovel
Three rotary blast hole drills, electric powered
One rotary blast hole drill, truck mounted
Nine crawler tractor dozers
One rubber tire tractor
Two road graders
Two 4000-gal sprinkler trucks
One 4000-gal fuel service truck
One lubricating service truck with enclosed heated body
One monorail repair truck with air compressor and welder
One powder truck
Two 44-passenger buses
Six 5-kw portable light plants
Two 3-ton lift trucks
One 65-ton diesel electric locomotive
One 120-ton diesel electric locomotive
One 100-ton transport
Three Jeeps
Three pick-up trucks

MAJOR TOOLS AND EQUIPMENT, SUB-REPAIR SHOPS

Welding and Plate Shop

Automatic crawler track welder
Automatic welder and positioner for track rollers, sprockets, etc.
Track press
Floor-stand grinders
Abrasive cut-off saw
Metal band saw
Drill press
Gas-fired forge
Anvil
Layout and work tables
Workbenches with vises
Electric welders
Acetylene welding equipment
Lockers for welding equipment and rods

Transmission, Differential, and Brake Repair Shop

Brake drum lathe
Brake drum riveter
100-ton hydraulic press
Floor-stand grinders
Parts cleaner
300-cu ft air compressor
Workbenches with vises
Transmission stands
Differential stands
Storage cabinets for transmission and differentials

Tool Room and Spare Parts

Special tools
Jigs
Fixtures
Gages
Indicators
Micrometers
Miscellaneous

Converter and Fuel Pump Shop

Parts cleaner
Fuel pump test stand
Injector test stand
Electric servicer
Converter test stand
Floor stand grinder
Steel workbenches and vises
Surface plates
Lapping plates
Work tables
Storage lockers
Electric oven

Motor Repair and Test Shop

28-in. drill press
16-in. sliding bed gap lathe
Parts cleaners
Floor-stand grinders
Surface grinder
Honing machine
Valve and seat facer
Boring bars
Rod aligner
Engine dynamometer and test stand
Engine stands
Workbenches and vises
Storage cabinets for special tools and gages

Electric Shop

Workbenches and vises
Floor-stand grinder
Vulcanizer for electric power cables
Storage cabinets for small electric parts

Grease Racks

Four sets of hoses and reels as follows:
Air
No. 20 motor oil
No. 30 motor oil
Hydraulic oil
Torque converter oil
No. 90 gear oil
No. 140 gear oil
Chassis grease

is filtered and dehydrated before going to the dispensing islands.

Fuel and oil can be delivered in railroad tank cars or by motor transport. One bulk fueling station is located south of the pump house and contains three filling units, with ticket printers, for loading the 4000-gal fuel tanker.

East of the pump house and north of the main garage are located three dispensing islands for servicing the trucks. Two of these are for diesel fuel and can service four trucks at a time. The other is for gasoline service. All pumps on the islands are equipped with meters and ticket printers. Complete records are kept of all fuel and oil received and dispensed to units.

The wash rack and paint shop are at the extreme north end of the yard. This is a steel frame building covered with corrugated iron inside and out and 1½-in. Fiberglas insulation sandwiched between. The wash rack is equipped with a high pressure combination steam cleaner and also a portable steam cleaner that can be taken into the pit to clean shovels, crawlers, and drills. Ample supplies of hot and cold water, steam, and solvents are available for all types of cleaning.

The paint shop is in the other half of the building and is provided with conventional spray-gun equipment. On the north side of the paint shop is a concrete apron where trucks are given a hosing down before being taken into the wash rack or grease pits. In warm weather washing with the cold water hose is all that is required to clean the trucks before they roll into grease pits or repair bays.

About 200 ft east of the main shop building is an outdoor electric power supply for the engine heaters in the parked trucks. The installation consists of nine 10-in. diam steel pipes set in concrete and extending 5 ft above ground.

Each of these pipes has four outlet boxes, one for each truck. Each plug-in cord furnishes power to two 2500-w heaters mounted on the engine of each truck. The arrangement will accommodate 36 trucks and assures easy starting during the winter.

The main shop was planned and equipped to provide for a good maintenance program. Such a program will accomplish the following.

- ► Permit handling more units.
- ► Increase efficiency of shop facilities and men.
- ► Reduce overall repair costs.
- ► Reduce downtime on expensive equipment.
- ► Allow control over labor and parts used.

Making such a system workable calls for a definite schedule of operations, assignment of these operations as a responsibility, and supervision.

Fast removal of complete assemblies such as engines, converters, and transmissions, whenever they show signs of distress, reduces damage to other parts of the equipment and reduces total downtime. Defective parts can then be sent to the sub-shops for complete overhaul.

Testing is highly important. Rebuilt engines for example, are run in on a dynamometer and all final adjustments of fuel pumps and injectors are made. When such a unit is placed in a truck, it is ready to operate at full power and the operator need not give it the normal careful break-in period.

Good departmental cooperation is very essential; each should know the problems of the other to keep cycle lags to a minimum. Every effort is made to promote harmony between the production and maintenance operations.

TUNGSTEN IN SEARLES LAKE

An unexploited reserve promises more tungsten than the total contained in all other known U. S. sources. The problem now: commercial recovery.

by L. GRAYDON CARPENTER and DONALD E. GARRETT

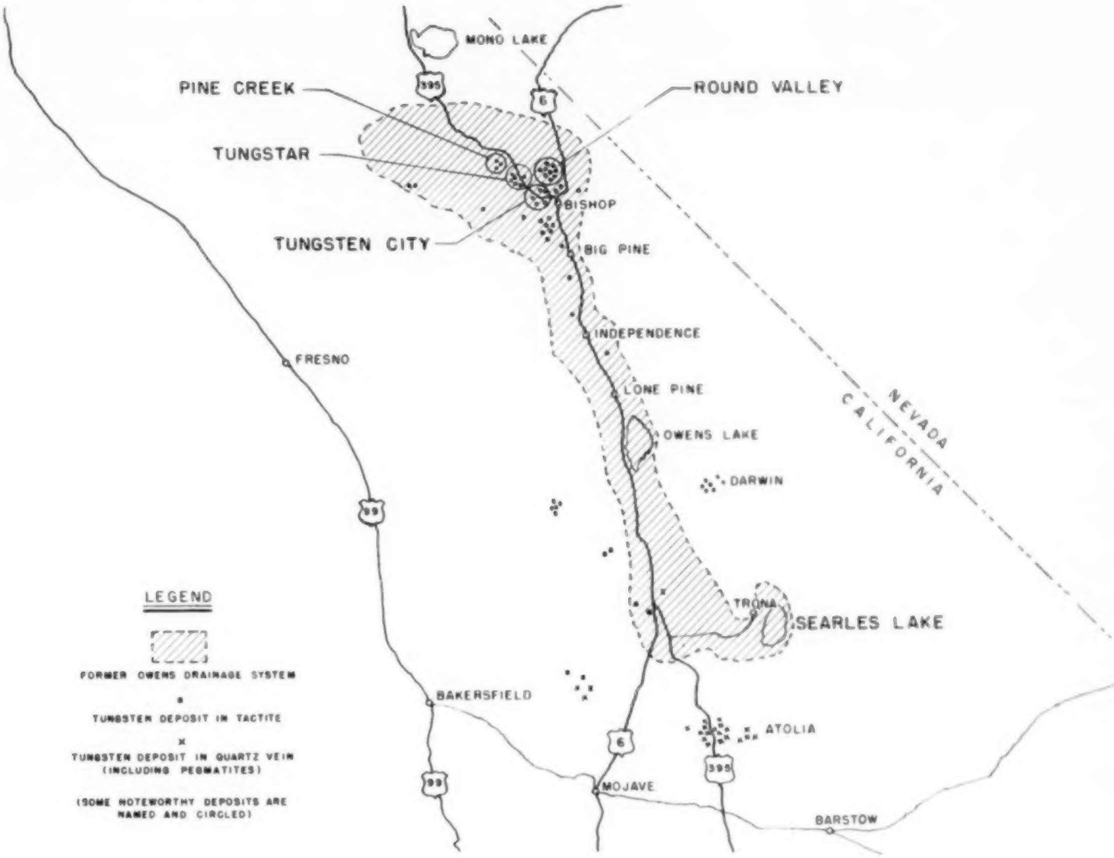


Fig. 1—Relation of tungsten deposits to former Owens drainage system.

Probably the largest single tungsten deposit in the U. S. is one that has yet to produce any tungsten; it is not even listed in tables showing U. S. reserves. This deposit is at Searles Lake, Calif., where the brine contains about 70 ppm WO_3. Small though the concentration is, the amount of brine in the lake is so great that it is estimated at 170 million lb of WO_3 (8.5 million units), equaling the total of all other known reserves in the U. S.

This dry lake is located in the basin formed by the Argus and Slate ranges in the northeastern corner of the Mojave Desert (Fig. 1). It consists of a massive salt body some 35 sq miles in extent, varying in thickness from 0 to 120 ft and containing more than 3 billion tons of salt. A central section of about 8 sq miles is exposed salt; the remaining area is overlain by mud. Except for the exposed sodium chloride (10 to 15 ft thick) the body consists of a very complex and heterogenous mixture of soluble crystals, containing at least eight major phases. It is only about 53 pct solid, the voids being occupied by a dense brine which is the raw material for two chemical companies operating in the valley.

The deposit is believed to be made up of salts present in the runoff waters from the last two ice stages. Searles Lakes was the third in a chain of five lakes that received waters from the Owens Valley drainage of the Sierra Nevada Mts. After leaving Owens Lake the waters could flood in succession into the Indian Wells, Searles, Panamint, and Death valleys (Fig. 2). During the first two ice stages of the Pleistocene Epoch most of the waters eventually arrived at Death Valley; consequently the lower 3300 ft of Searles Valley contain only mud

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Table I. Typical Upper Structure Brine Analysis from Searles Lake

Constituent	Upper Structure Brine, Wt Pct
$NaCl$	16.10
Na_2SO_4	6.75
KCl	4.90
Na_2CO_3	4.75
$Na_2B_4O_7$	1.58
$NaHCO_3$	0.15
Na_3PO_4	0.14
Na_2S	0.12
Na_3AsO_4	0.05
Br^-	0.085
Li_2O	0.018
WO_3	0.007
I^-	0.003
F^-	0.002
H_2O (by diff)	65.34

and insoluble salts with thin and scattered seams of sodium chloride, trona, and nahcolite. Following the second ice stage, however, there was volcanic activity in the Mono area and in the White Mts. (both extending into the Owens drainage area). Considerable volcanic ash and debris resulted, and numerous hot springs were formed. This greatly enriched the mineral content of the runoff waters.

In the succeeding glacial periods, the Tahoe and the Tioga, there appears to have been little, if any, overflow of waters from Searles Lake, so that it became the repository for most of the salts. During the Tahoe period about 40 ft of mixed salts were deposited, with six mud seams from 2 in. to 2 ft banded through the deposit. Later, in the Tioga period, about 14 ft of mud were deposited and then about 80 ft of salts. The top of the 14-ft mud seam is approximately 10,000 years old and the bottom 24,000 years, as indicated by carbon 14 dating.

The complexity of the brine mixture within the salt beds is shown by the list of major constituents in Table I. The tungsten in the brine probably exists as a large heteropoly ion, $(M_xW_yO_z)^{n-}$, where M may be boron, arsenic, or phosphorus and $y{:}x$ may be numbers between 6 and 12. Such ions are very soluble in water but will form insoluble precipitates with complex organic compounds, such as proteins and various alkaloids.[1] The concentration in the brine is only 0.005 to 0.008 pct WO_3, and 0.2 to 0.25 pct in some of the plant liquors—well under the saturation concentrations that would allow a solid phase of tungsten to crystallize out.

The tungsten in the Searles Lake deposit probably resulted from a combination of the leaching of minerals and from the tungsten content of waters from hot springs. Many known tungsten deposits are within the drainage basins (Fig. 1) of the Owens, Indian Wells, and Searles valleys. The largest is in the Bishop district, principally the Pine Creek mine of U. S. Vanadium Corp. There are known reserves of more than one million units of tungsten in the ore, which generally contains 0.3 to 1 pct WO_3, averaging 0.45 pct. In a portion of the ore zone molybdenum disulfide is about 1 pct and copper 0.2 pct. Silver is also present. Other mines in the area, such as the Black Rock mines, contain ore averaging 0.6 pct WO_3.

One of the country's largest producers has been the Atolia district somewhat to the south, although its reserves are now small. This deposit differs from most of the others associated with the Sierra Nevada batholith in that the scheelite is associated with quartz and quartz monzonite. Some of the deposits were of high grade, averaging 4 pct WO_3. About 10 pct of production was from placer deposits, where concentration was much lower.

Most of these California deposits are scheelite, generally in tactite, a rock formed by the alteration of limestone or marble by intrusive granitic magma. It is thought[2] that the tungsten deposits were formed following the intrusion of quartz monzonite and granite into overlying sedimentary deposits. Mineralizing solutions were rich in silicon, aluminum, and ferric iron but poor in sulfur. They carried tungsten, which was precipitated by the limestone at the contact zone as calcium tungstate (scheelite). The solutions may well have been at high temperatures, perhaps even in the gaseous state. The deposits were thus of the contact-metamorphic type, and as would be expected from this mode of deposition, the tungsten content of the ore is generally low.

The present-day Keough hot spring possibly resembles the original tungsten-depositing waters, for it analyzes 0.3 ppm WO_3.[4] It is very likely that this hot spring was formed during the last period of volcanic activity in the area and can be assumed to have played a major part in supplying tungsten to Searles Lake. Assuming a dilution factor of 300 for the Keough to total Owens River waters, the total tungsten concentration (from that source) would be about one part per billion. When the Searles Lake ratio of total sodium chloride to total tungsten was compared with the assumed average sodium chloride content of the Owens River waters (the 1908 to 1958 average was used), the total tungsten content of the ice age Owens River waters was calculated to be 4 parts per billion. Considering that the saturation value of scheelite in water is about 0.2 g per 100 g of water, and taking into account the large amount of exposed tungsten in the drainage basins, the additional 3 parts per billion in excess of that from the one hot spring is reasonable. The comparatively high solubility of scheelite and the complex tungstates would indicate that no tungsten precipitated or crystallized during evaporation of the ancient lakes and that essentially all the tungsten originally present is now in the brines of Searles Lake.

Tungsten was first recognized in Searles Lake brines in 1937 when the scale clogging a condenser line was analyzed. Since then considerable study has been directed toward developing an economic process to recover tungsten from the plant liquors. It is estimated that processing the higher strength plant liquor would recover approximately 1000 lb of tungsten per day. Processing the brine directly from the lake, of course, would produce a larger amount, but recovery would be much more costly, since concentration would be thirtyfold less.

Solvent extraction or ion exchange methods would appear attractive, in view of the uranium industry's success with low concentrations of metal ion in concentrated salt or acid solutions. Unfortunately such processes have not been effective with Trona liquors. A large number of ion exchange resins have been tested, including chelating resins specific for tungsten, without any removal of tungsten from the brine. Similar tests with a wide variety of potential solvents or complexing agent-solvent mixtures have been made with the same negative results. This would appear to indicate that the tungsten complex exists as a low-valenced ion and that it is relatively stable and not easily dissociated.

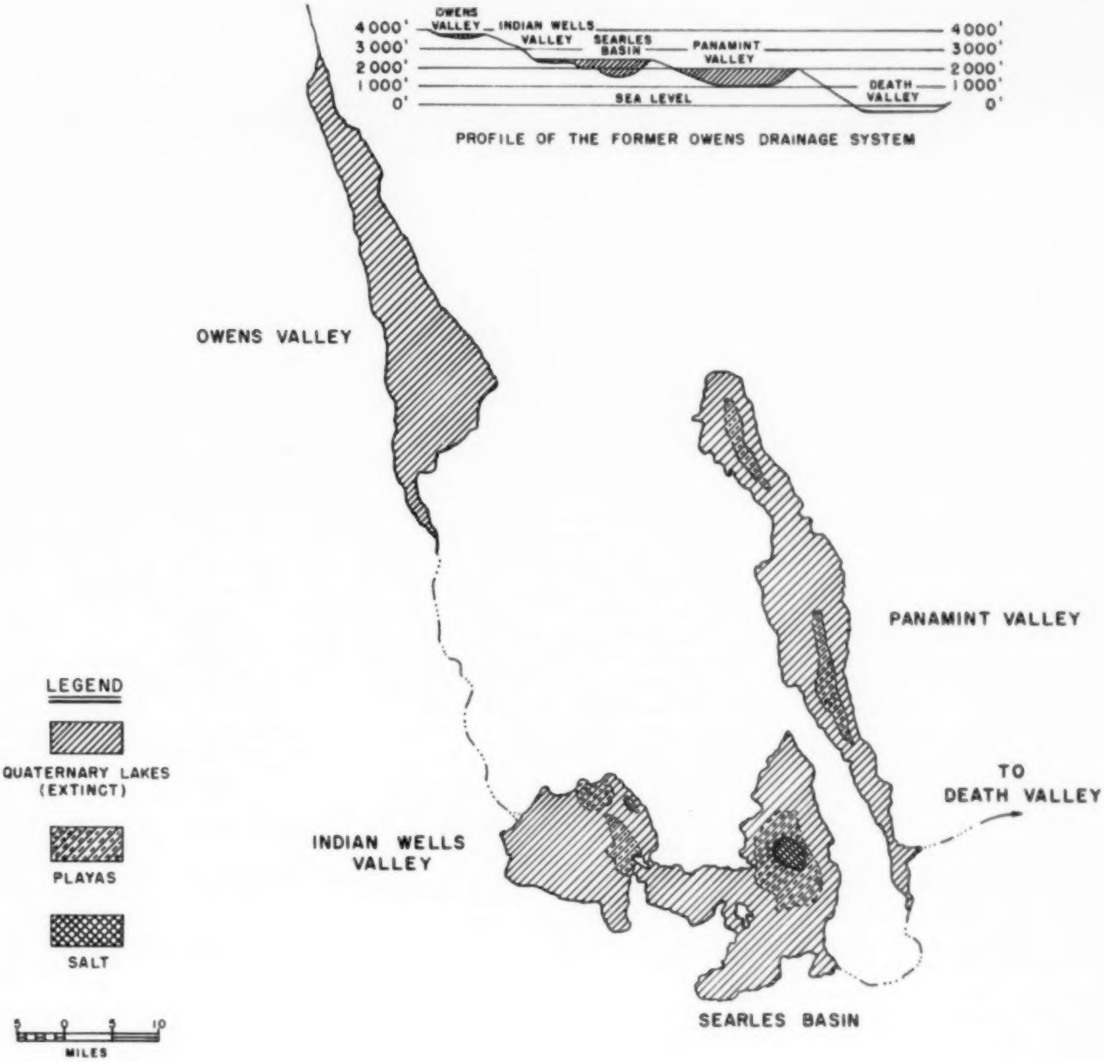

Fig. 2—The former Owens drainage system.

Tungstic acid is quite insoluble, so by suitable acidification of the liquors, tungsten can be precipitated. It is possible to acidify the liquor directly or to use an indirect method, such as electrolysis to generate chlorine, thus acidifying the solution and simultaneously oxidizing the tungsten. The tungsten recovered by acidification is very impure, even after the soluble salts have been removed, containing large quantities of arsenic, phosphorus, and molybdenum. An extensive purification step is required, further adding to the expense and impractibility of such a method.

If a precipitant such as iron or blood meal is used, less acidification is needed and the process is cheaper. However, considerable acid is still necessary, and the product is again very impure. Although they are an improvement on direct acidification, these precipitants do not provide an economical process.

A much more promising approach has recently been studied involving precipitation of tungsten with an organic complexing agent. An essentially complete precipitation of tungsten may be made from liquor at a pH that can be reached by the comparatively easy device of carbonating the liquor with flue gas. The precipitate is considerably less contaminated than by previous methods, but purification is still required if the tungsten is to be of filament grade. The economics of this process are not yet defined, and it is not known whether a commercial process will result, but the studies have added to the knowledge of tungsten as it occurs at Searles Lake. It is believed that eventually this tungsten reserve, the country's largest, will be at least partially recovered.

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by M. C. CHANG and JOHN DASHER

MINERAL DRESSING FUNDAMENTALS

The Crucible mine in Pennsylvania, operating on Pittsburgh seam coal, is rated at 5000 tpd. The washing plant, built in 1943, is rated at about 400 tph, using hydroseparator boxes to wash the coarse coal (4 x 5/16 in.) and hydrotator units for the fine coal (5/16 x 0).

It is generally acknowledged that gravity concentration equipment cannot be used efficiently to beneficiate coal finer than about 65 mesh. Before the present flotation circuit was installed, Crucible recovered its fine coal slurry, predominantly −24 mesh material, by passing it through a battery of five 14-in. and one hundred and ten 3-in. Heyl & Patterson cyclones in parallel with a 75-ft Dorr thickener. About two thirds went to the cyclones and one third to the thickener. Underflows from cyclones and thickener were filtered, respectively, in one 12-disk Eimco and two 4-disk Oliver filters fitted with stainless steel cloth. The resulting cakes were added to the coarse metallurgical coal. The filtrates, overflow from the 3-in. cyclones, and thickener overflow were recycled to the coarse coal plant.

The three major problems encountered in this system became acute when the mine switched from track-mounted machinery to the more economical off-track equipment. The quality and uniformity of product declined, too much slime accumulated in the circuit, and waste pond disposal became highly impracticable.

Lowering of Quality and Uniformity: Since the filter cake was not beneficiated in any way, addition of this material, which contained about 18 pct ash, would certainly lower the quality of metallurgical coal. Difficulties of handling the high-ash, high-moisture cake were compounded by the lack of adequate blending facilities at the mine, making it impossible to maintain uniform feed for the coke ovens. For a two-month test period, to verify the source of trouble, all filter cake was excluded from the metallurgical coal. The excluded filter cake was stocked in the yard.

Slime Buildup: The moisture retained in the coarse coal products and the filter cake provided daily the only means of slime bleed from the circuit. As this bleed was not enough to offset the incoming fines in the raw coal feed, there was excessive buildup of slimes. To operate at all, it was necessary to purge the system each weekend by pumping all the solids from the thickener to a waste pond. The slimes built up during the week progressively increased the apparent viscosity and density of the circulating plant water, so that hydroseparators and hydrotator units had to be constantly adjusted. Uniformity, yield, and quality of the metallurgical coal suffered in consequence, and the filter cake ash and moisture contents increased from Monday to Friday.

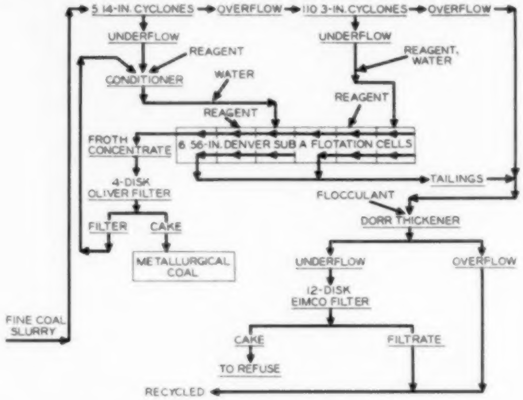


Fig. 1—The flowsheet for Crucible's fine coal plant.

Limited Waste Disposal: Location of the plant does not permit unlimited waste pond area. Maintenance of a waste pond for the purpose of emptying the thickener during weekends became a costly burden that could not be tolerated.

HOW THE PROBLEMS WERE SOLVED

The flowsheet (Fig. 1) chosen to overcome these difficulties is considered the one most economical to install and operate. In a preliminary change made before the flotation circuit was installed, the cyclones and thickener were first operated in series, the overflow from the 3-in. cyclones, after flocculation, being discharged into the thickener and the thick-

Table I. Average Analysis of Filter Cakes

Date	Ash, Pct	Sulfur, Pct	Remarks
January 1956	18.41	—	Old practice
September 1956	15.80	2.29	Cyclone and thickener in series, before flotation
August 1957	8.17	2.02	After flotation

Table II. Estimated Cost of Beneficiating Filter Cake by Flotation

Fixed Charges, Based on New Equipment		Clean Filter Cake, Cost Per Ton, $
Taxes, etc., at 2 pct of $50,000		0.013
Depreciation at 10 pct of $50,000		0.063
	Total	0.076
Operating Costs:		
Power at $0.01 per kw-hr		0.030
Reagent at $0.17 per lb and 0.1 lb per ton		0.017
Maintenance at 2 pct of $50,000		0.013
	Total	0.060

M. C. CHANG, Member AIME, is Research Mineral Engineer and J. DASHER, Member AIME, is Research Supervisor, Central Research Laboratory, Crucible Steel Co. of America, Pittsburgh. TP 4795F. Manuscript, March 31, 1958. New York Meeting, February 1958. AIME Trans., Vol. 214, 1959.

APPLIED TO THE FINE COAL PROBLEM

ener underflow being pumped to a temporary waste pond.

Series Operation of Cyclones and Thickener with Flocculation of Thickener Feed: The flowsheet was first arranged so that all fine coal slurry was classified in the cyclones, only the final overflow discharging into the thickener. In this operation the cyclones removed the relatively coarser material and rejected the high-ash fines. Cyclone performance is shown by the size and weight distribution of feed and products given in Fig. 2. About 25 pct of the solids fed into the cyclones was rejected in the final overflow. Subsieve sizing of solids in the final overflow showed that 92 pct was finer than 600 mesh. In the 3-in. cyclone underflow 88 pct of the material was coarser than 600 mesh.

To provide a clear overflow for recycle, the thickener feed was flocculated with causticized potato starch. This change of circuit, made before flotation, somewhat improved the quality of product in the cyclone underflows.

Preparing the Feed: Rarely used in soft coal beneficiation, flotation in this field has generally been considered unfeasible and uneconomical, an opinion based on results from plants where the process has been misapplied. When minerals are coated, for example, by excessive slimes, surface properties of the various phases are equalized, making it impossible to separate by flotation. The high specific surface of slimes demands excessive reagent and causes considerable difficulty in handling the froth product, and the fine coal slurry contains much fine clay. Success in flotation of coal depends on proper desliming of feed.

It also depends on uniformity of feed. The flotation rate of a mineral finer than 100 mesh usually decreases with decreasing size. When a feed consists of discrete particles of different minerals with wide differences in floatability, particles in a wider range of sizes can be separated with reasonable efficiency. Fine coal particles, however, are middling, that is, each contains some coal and some ash. To separate those which are mostly ash from those which are mostly coal, it is necessary not only to minimize the proportion of slimes, but also to have a feed of reasonably uniform size. Only under these conditions will it be possible to achieve good separation, reagent and power economy, and efficient use of equipment.

The existing 14-in. and 3-in. cyclones at the Crucible plant could be used to deslime and classify the flotation feed. The underflows, which consisted of different size ranges, were floated separately.

Operation: Each of the cyclone underflows was floated in three No. 30 Denver Sub A cells. The bank of six cells can handle 20 tons of clean coal per hr.

Feed is maintained at about 20 pct solids. The reagent in the 14-in. cyclone underflow is Aerofroth No. 73 and in the 3-in. cyclone underflow methyl isobutyl carbinol. Average reagent consumption is about 0.1 lb per ton of clean filter cake.

The froth product filters readily and is being handled in a 4-disk Oliver filter. The same material, unbeneficiated, required about three times as much filter area. The tailing is discharged into the thickener for dewatering and disposal.

Performance: The improvement in quality of the filter cake by flotation can best be described by comparing the average analysis of the various filter cakes given in Table I. Flotation reduced the ash and sulfur in the cake; rejection of ash, however, is more effective than that of sulfur. Weight recoveries are 8 to 90 pct for the 14-in. cyclone underflow and 80 to 85 pct for the 3-in. underflow.

Cost: The estimated cost of beneficiating the filter cake by flotation is given in Table II. Total cost is about 14¢ per ton of clean filter cake. Since this product is about 8 pct of the total tonnage produced, the additional cost for cleaning the filter cake is a little more than 1¢ per ton of total product. Labor cost is not included in the estimate, since no extra labor was required to operate the flotation units.

Efficiency of separation by size and ash distribution on samples of the flotation feeds, concentrates, and tailings is shown in Figs. 3 and 4 for the 14-in.

Table III. Comparison of Product Quality

Date	Mine Coal Analysis: Ash, Pct	Mine Coal Analysis: Sulfur, Pct	Remarks
January 1956	8.74	1.60	Previous practice, filter cake to barges
April 1956	8.07	1.61	Previous practice, filter cake excluded
September 1956	8.65	1.49	New practice, before flotation
August 1957	7.44	1.35	New practice, after flotation

Table IV. Comparison of Product Uniformity

Date	Coke Plant Coal Analysis: Ash, Pct	Coke Plant Coal Analysis: Variant	Coke Plant Coal Analysis: Mean Deviation	Remarks
January 1956	9.11	0.79	0.66	Previous practice, filter cake to barges
April 1956	7.82	0.30	0.42	Previous practice, filter cake excluded
September 1956	8.62	0.56	0.60	New practice, before flotation
August 1957	7.30	0.25	0.42	New practice, after flotation

Table V. Results of Geisler Plastometer Test

Sample	Start, °C	Final, °C	Range, °C	Max, °C	Fluidity, DDPM
Clean production filter cake	386	473	87	432	55,550
Unclean stock filter cake	385	469	84	430	37,500
Clean stock filter cake	368	466	98	429	41,700
Run-of-mine washed coal	381	478	97	438	33,400

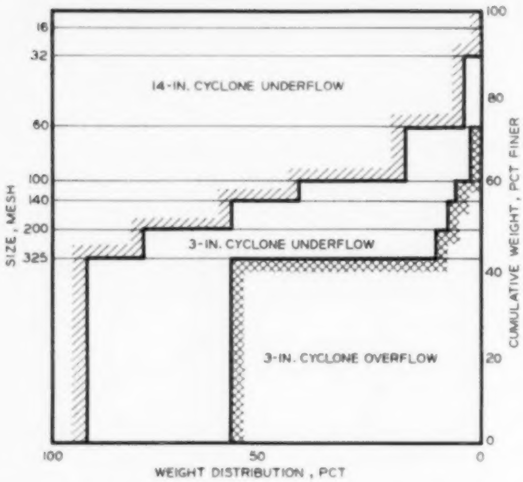


Fig. 2—The size and weight distribution of cyclone feed and products. Approximately 25 pct of the solids fed into the cyclones was rejected in the final overflow.

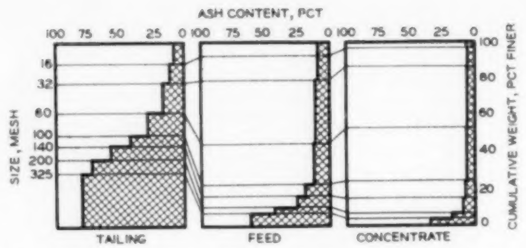


Fig. 3—Size and ash distribution, 14-in. cyclone underflow.

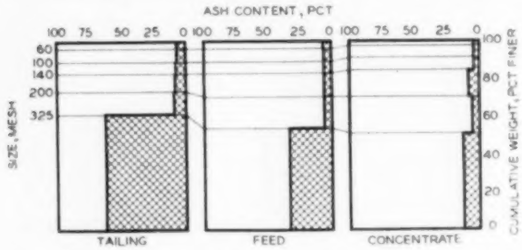


Fig. 4—Size and ash distribution, 3-in. cyclone underflow.

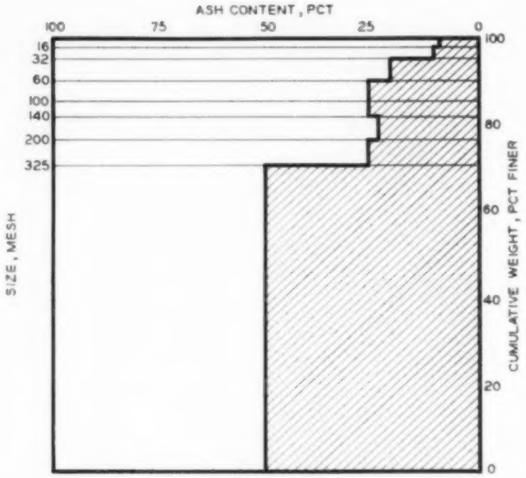


Fig. 5—Size and ash distribution of the fine refuse.

and 3-in. cyclone underflows, respectively. It may be noted that a good separation (concentrate containing less than about 10 pct ash) is obtained in the 14-in. cyclone underflow circuit only above 200 mesh; in the 3-in. cyclone underflow circuit, a good separation is obtained on the −325 mesh (but deslimed) fraction. This verifies the necessity of classifying feed.

Filtration of Thickener Underflow: Thickener feed now consists of the 3-in. cyclone overflow and the flotation tailings. For flocculation the starch consumption is approximately 0.6 lb per ton of solids. Thickener underflow is filtered in the 12-disk Eimco filter fitted with polyethylene bags. The cake, containing 40 to 50 pct ash and 30 to 35 pct moisture, is combined with the plant refuse for disposal via aerial tram buckets to the gob pile located across the river. This closed circuit operation of the plant eliminates the need for a waste pond, except for emergencies, and helps remedy the problem of stream pollution.

Size and ash distribution of solids in the filter cake refuse is shown in Fig. 5.

RESULTS EVALUATED

Product Quality: The improved quality of products resulting from these changes is shown by the average analysis of the various products given in Table III. Quality of the metallurgical coal has been significantly improved, as shown by the lower ash and sulfur contents.

Product Uniformity: At Midland a barge is unloaded by a clamshell, which dumps the coal in a hopper for discharge onto a conveyor belt to the coke plant. Any variation in uniformity of product would be reflected in the daily Midland analysis of the coal. The increased uniformity of product quality is evident from Table IV, which gives the average calculated variant and mean deviation under the various washer operations.

Geisler Plastometer Tests: Results of the Geisler plastometer test on the clean production filter cake; the stock filter cakes, clean and unclean; and the run-of-mine washed coal are given in Table V. It is interesting to note that all of the filter cakes tested give higher fluidity reading than the run-of-mine washed coal.

Data presented in this article were based largely on samples taken by the staff of Heyl & Patterson. The authors express their appreciation to Crucible Steel Co. of America for permission to publish this information.

Discussion of this article sent (2 copies) to AIME before April 30, 1959, will be published in MINING ENGINEERING.

A DECADE OF DEVELOPMENT IN OVERVOLTAGE SURVEYING

by ROBERT W. BALDWIN

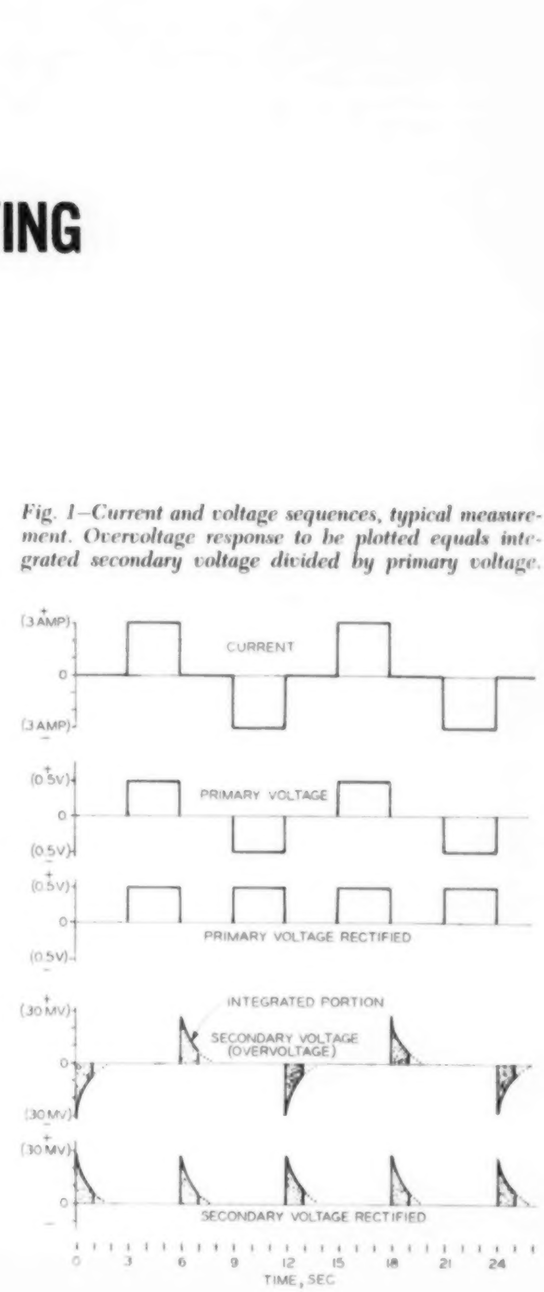
As used in geophysical exploration, the term *overvoltage* applies to secondary voltages set up by a current into the earth which decay when the current is interrupted. These secondary effects may be measured by pick-up electrodes. The term *induced polarization* has often been employed to describe this same phenomenon. In its own operations Newmont Exploration Ltd. commonly uses the word *pulse*.

The basis of this method in prospecting is that metallic particles, sulfides in particular, give a high response, whereas barren rock, with certain exceptions, gives a low response. Overvoltage has been tried in searching for many types of mineral occurrence but has been most successful in outlining the widespread disseminated mineralization associated with porphyry coppers.

History:[1] Newmont Mining Corp. has been interested in overvoltage since 1946, when Radio Frequency Laboratories of Boonton, N. J., drew the company's attention to phenomena observed in the laboratory. At the instigation of A. A. Brant further model studies were undertaken, and the first tests were performed in 1947. Tests at San Manuel, Ariz., in 1948 were very encouraging, clearly demonstrating that the method could be used to distinguish scattered sulfides at depth. H. O. Seigel followed up the San Manuel work with a study to determine the phenomena involved.[2]

R. W. BALDWIN, Member AIME, is with the Geophysical Department, Newmont Exploration Ltd., Danbury, Conn. TP 4793L. Manuscript, June 25, 1958. New York Meeting, February 1958. AIME Trans., Vol. 214, 1959.

Fig. 1—Current and voltage sequences, typical measurement. Overvoltage response to be plotted equals integrated secondary voltage divided by primary voltage.



Further field experiments took place at Jerome, Ariz., in 1949-1950. Since 1950 this method has been a standard prospecting tool of Newmont Exploration Ltd. Overvoltage surveys have been carried out in the U. S., Canada, Latin America, and Africa. Field equipment has been constantly improved.

Concurrent with field exploration, theoretical and experimental investigations were pursued at Jerome. H. O. Seigel, J. R. Wait, V. Mayper, E. H. Bratnober, and L. S. Collett were notable contributors. Work at the Jerome laboratories included:

1) Study of the phenomena involved, with extensive investigation into the causes of background nonsulfide effects.

2) Study of the possibilities of taking induced polarization measurements with low-frequency alternating current instead of pulsed direct current.

3) Mathematical development of type curves showing the anomalies to be expected from mineralized bodies of various shapes and sizes under varying depths and conditions of cover.

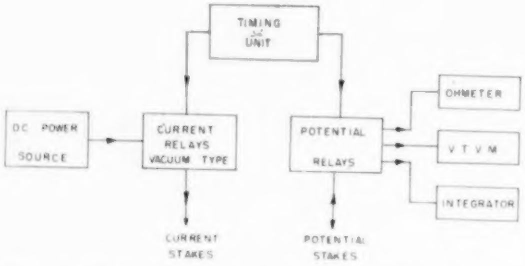


Fig. 2—Block diagram of typical field equipment.

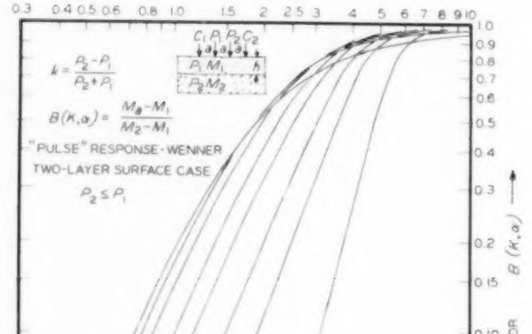


Fig. 3—Typical theoretical overvoltage response curves.

4) Laboratory testing of rock samples, study of the form of overvoltage decay and the a-c response for various types and sizes of mineral particles, and model orebody studies.

Operational Methods: The overvoltage method requires direct connection to the ground, by means of two current electrodes and two potential electrodes. Field methods are thus similar to those of resistivity surveys. Various electrode arrays have been used; electrode spacings are chosen according to the type of target and expected depth. Spacings as wide as 1500 ft have been regularly employed. In laboratory work also, four direct connections must be made to the specimen or model.

Fig. 1 illustrates, in idealized form, the sequences encountered in a typical d-c overvoltage measurement.* While the current is on there is a primary voltage across the potential electrodes which may be measured with a vacuum tube voltmeter—a simple resistivity measurement. On cessation of current (allowing 10 to 15 milliseconds for inductive and capacitive coupling effects to disappear) the decaying secondary voltage or overvoltage appears at the potential electrodes. This decay curve may be presented on an oscilloscope and photographed—the procedure in many laboratory experiments. Field practice is to integrate the decay voltage over an interval following current cessation. Common operating times are 3 sec of current pulse and 1 sec of integrating time. To obtain a reading the integrated secondary voltage is divided by the primary voltage. The units are then millivolt-seconds per volt.

* The voltage and current values quoted are samples to indicate an order of magnitude.

In practice, of course, not just one pulse of current is applied but a succession of pulses as shown, every second pulse being of reversed polarity. Rectifying relays are provided so that the primary and secondary voltages always read positively.

Field Equipment:[3] Fig. 2 is a block diagram of typical field equipment. The heart of the equipment is the timing unit, which controls both current switching and the connections of potential electrodes to the vacuum tube voltmeter for primary voltage and to the integrator for secondary voltage measurement. Two types of timing units have been employed: the first electronic, using multivibrators, and the second mechanical, using a constant-speed motor and cam-operated switches. The integrating device is a General Electric fluxmeter, model 32C248. The d-c power supply has usually consisted of a gasoline-motor a-c generator followed by a high-voltage d-c rectifier unit. The smaller units (order of 1000 to 1500 w) are relatively mobile and have been transported by burros; the larger units (up to 25,000 w) are mounted in heavy-duty trucks.

Most field equipment was designed and constructed in the Jerome laboratories by A.W. Love, K. E. Ruddock, and W. E. Bell.

Type Curves:[4] H. O. Seigel has developed mathematical expressions for the overvoltage response to be expected from mineralized bodies of various geometric forms. The analysis is equally applicable if the source of overvoltage effects is not mineralization. Seigel uses an electrodynamic model of overvoltage which considers the effect of resistivity contrasts within the region of measurement on both primary and secondary fields. His basic postulate is that the action of the primary field sets up a volume distribution of current dipoles—all antiparallel to the primary field—whose moment equals the product

of the primary current density and a mineralization*

* The term *mineralization* is understood to include other sources of overvoltage effects.

factor which is a property of the medium. He then develops a procedure for calculating overvoltage responses from associated resistivity curves by weighting the overvoltage contribution of any medium according to the logarithmic derivative of apparent resistivity with respect to the resistivity of that medium.

Mathematically,

$$M_a = \sum_i M_i \frac{\partial \log \rho_a}{\partial \log \rho_i}$$

where M_i and ρ_i are the *mineralization* factor and apparent resistivity of the ith medium, M_a and ρ_a are the overvoltage response and apparent resistivity at the point of measurement and $\sum_i$ represents a summing of the terms for all media.

Where there are only two media concerned the above formula reduces to

$$\frac{M_a - M_1}{M_2 - M_1} = \frac{\delta \log \rho_a}{\delta \log \rho_a}$$

where the subscripts 1 and 2 refer to media 1 and 2.

An important approximation of overvoltage surveys is the two-layer case. This assumes a horizontal layer of barren material overlying an infinite layer of mineralized material. The overvoltage responses have been derived directly from the well known resistivity two-layer formula. Fig. 3 gives the type curves when the lower layer has the lower resistivity. The abcissa is relative electrode spacing (i.e., in terms of thickness of top layer) and the ordinate, in effect, indicates what proportion of the lower layer mineralization factor should appear in the observed reading. The different curves are for different resistivity contrast conditions. Note that the plotting is logarithmic. Examples of the use of these curves are given in the field results to follow.

Phenomenological Theory:[5] To account for overvoltage effects, J. R. Wait has proposed the following theoretical model:

Each conducting particle is considered to be coated with a thin dielectric film that poses a block action to current flow into the particle. Thus the action at the interface of each particle is somewhat comparable to that of a lossy condensor, and any ground exhibiting an overvoltage response may be considered to contain in effect a large number of tiny condensers. It should be noted, however, that the dielectric constant of these condensers may vary with frequency.

Wait applied his model theory to predict the form of the decay curve and its variation with particle size. His predictions have been borne out by laboratory experiments. Some typical results are shown in Fig. 4. The tests were performed on a compact mixture of 98 pct andesite and 2 pct pyrite particles, plus a weak electrolyte. Different samples contained different sizes of pyrite particles, ranging from 0.25 to 12-mm diam. Duration of current pulse was 1 sec. Primary voltage was the same in all cases. Note that the time scale is logarithmic. It will be observed that decay is more rapid with the smaller sulfide particles. It can also be noted that at any time following the cessation of current there is an optimum particle size for which the decay voltage is maximum.

A-C Overvoltage Methods:[6] As is perhaps suggested by the condenser analogy mentioned above,

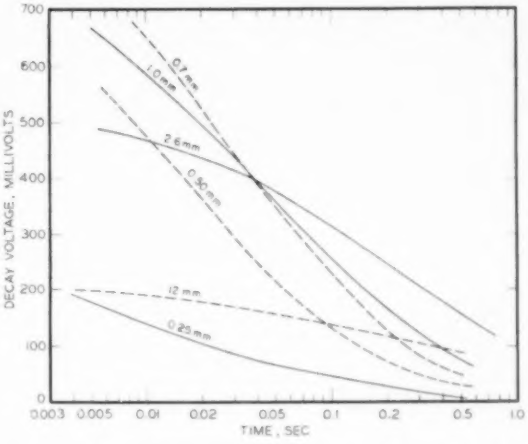


Fig. 4–Observed decay voltage $e(t)$ as a function of time. For $V = 15$ volts, $v = 0.02$ and $\sigma = 5x10^{-3}$ mhos/m. This graph and Fig. 5 are examples of extensive overvoltage experiments at Newmont's laboratories in Jerome, Ariz. Fig. 4 illustrates work in transient domain, Fig. 5 work in frequency domian.

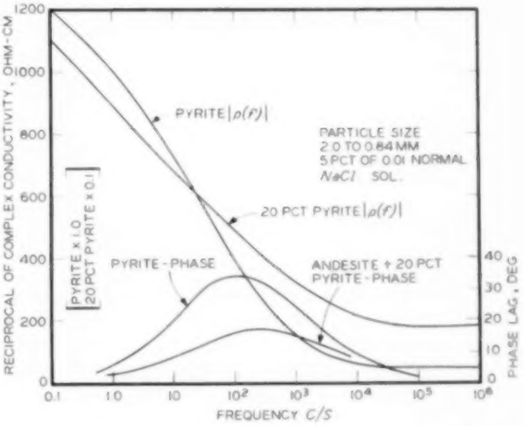


Fig. 5–Variation of complex conductivity with frequency. From experiments at Newmont laboratories.

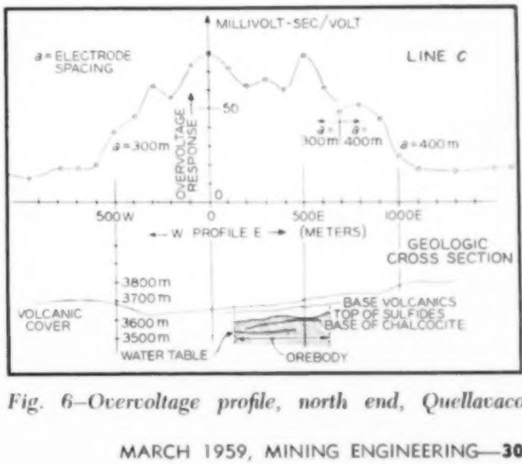


Fig. 6–Overvoltage profile, north end, Quellavaco.

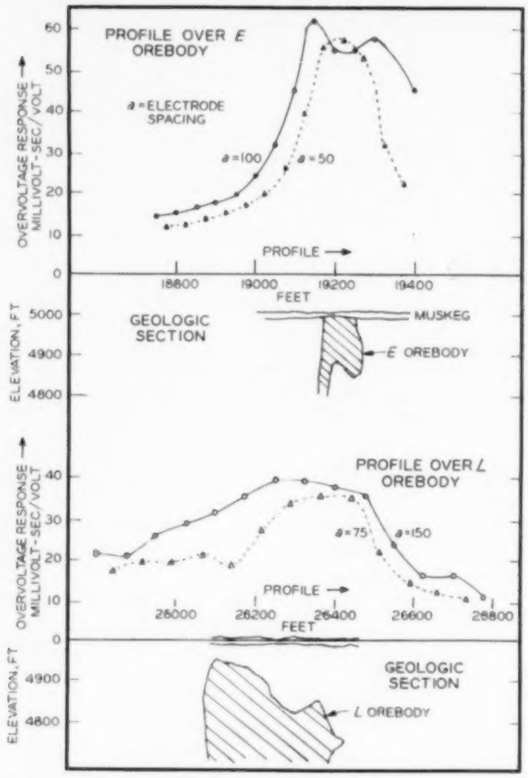


Fig. 7—Overvoltage profiles at Lynn Lake, Manitoba.

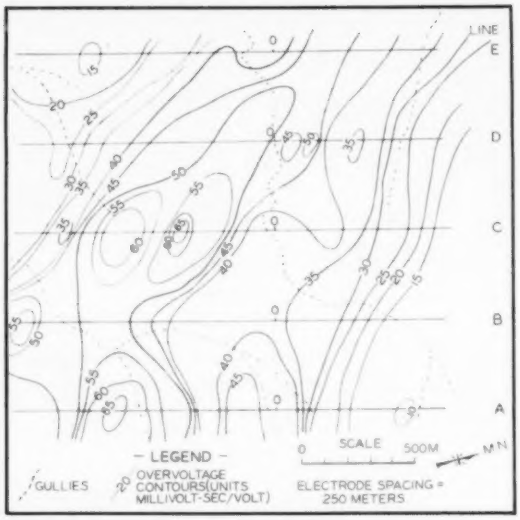


Fig. 8—Copper prospect, Peru. Overvoltage contours here directly outline distribution of sulfides.

the overvoltage phenomena may be measured in the frequency domain instead of in the transient domain, that is, by applying alternating current instead of pulsed direct current. The earth in general has a complex impedance in which the d-c resistivity is a pure resistive component and the overvoltage contributes a somewhat complicated combination of capacitance and resistance. The complex impedance and the phase angle vary with frequency. This variation is especially pronounced in the case of sulfides.

Results of some complex impedance measurements in the laboratory are shown in Fig. 5. Complex impedance and phase angle for pyrite and for pyrite in andesite particles are plotted against log frequency. The maximum slope of the impedance curve occurs at that frequency at which phase angle is a maximum. In comparison, impedance vs frequency curves for barren rock material (over the frequency range up to the order of several hundred cycles) are almost flat and the phase angle remains low.

It should be noted that a-c overvoltage measurements should be made in the low frequency range where electromagnetic propagation effects are negligible. Caution should also be taken to avoid excessive line coupling between the current and potential circuits. Probably several tens of cycles is about the upper frequency limit for operations in the field.

Wait has demonstrated the relation between the response in the frequency domain and that in the transient domain. From experimentally observed frequency response data he derived the overvoltage decay curve to be expected following a pulse of direct current. The agreement with the experimentally observed decay curve was excellent.

Field Results:[7] To date pulsed d-c methods have been used in field exploration. The technique of measurement is described above under Operational Methods and Field Equipment.

To repeat, the basis of the overvoltage method as a prospecting device is that metallic particles, especially sulfides, give a high response, whereas barren rock, with certain exceptions, gives a low response.

In the earlier days it was not realized that barren rock could display a considerable range of response, and minor anomalies of less than 50 pct of background were deemed evidence of sulfides. At Jerome, Ariz., anomalies of this order were found to be caused by certain portions of the Pre-Cambrian basement beneath the Palaeozoic cover. At the present time overvoltage readings of two to three times background are usually necessary to excite interest. Even then it must be recognized that some anomalies may have causes other than sulfides.

In overvoltage surveys results fall into four classes:

1) No significant anomalies.
2) Anomalies due to economic sulfides.
3) Anomalies due to noneconomic sulfides.
4) Anomalies due to nonsulfides.

Groups 2 and 3 above may both be considered geophysical successes if not exploration successes. The ratio of noneconomic to economic mineralization disclosed is certainly no worse than for other geophysical methods. The chief villain has been disseminated pyrite. Many porphyry copper deposits have a surrounding halo of disseminated pyrite, and the zone of maximum sulfides is not necessarily the zone of maximum copper.

While there have been a few striking examples of nonsulfide anomalies, most major anomalies have been explained by sulfides. For example, in almost four years of work in Peru, only one recommended

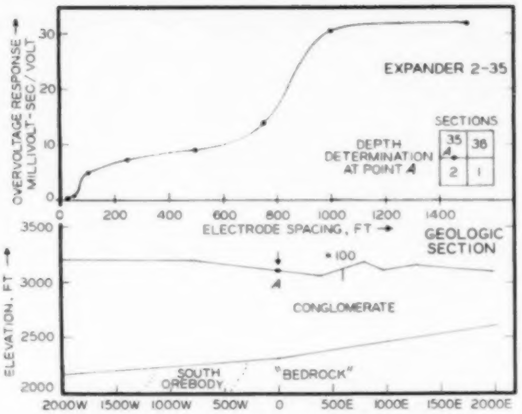


Fig. 9—Detection of deep mineralization is possible at San Manuel by use of large electrode spacings.

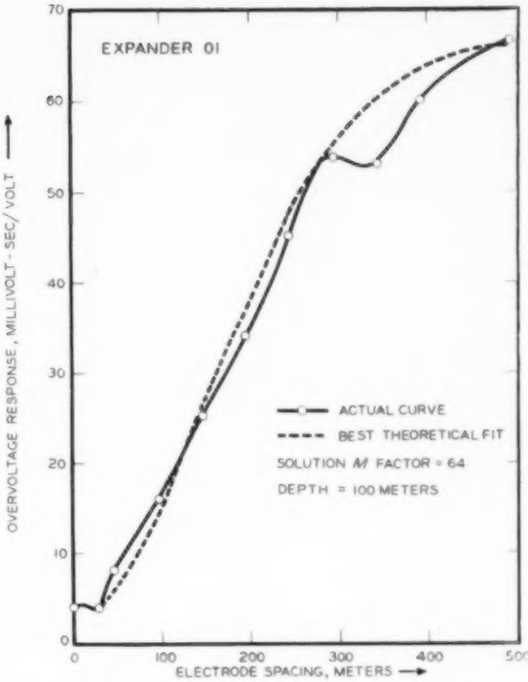


Fig. 10—Mineralization and depth, Cuajone.

drillhole completely failed to find a reasonable quantity of sulfides.

Over a disseminated sulfide deposit the anomalous overvoltage response (i.e., in addition to the rock background) will depend on:

1) The percentage by volume of sulfides.
2) The geometry of the deposit with respect to surface and the electrode array in use. Geometry thus includes size and depth below surface.
3) The resistivity contrast conditions between the sulfide zone and the cover and surroundings.

In any one area the overvoltage response of a mineralized zone has been found to vary more or less directly with the percent of volume of sulfides for moderate percentages of sulfides. It is not safe, however, to project from one area and type of mineral occurrence to another.

A fair number of the examples to follow were obtained over known or later proven orebodies. In attacking any new area, it has been the general policy to test over known mineralization first, where possible, and work out from there, so that the type of anomaly to be sought is known.

Fig. 6 shows an overvoltage profile over the north end of the orebody at Quellaveco, Peru. The ore zone is covered by about 40 meters of postmineral volcanics, and depth to sulfides is from 60 to 100 meters. The orebody is well detected; however, it is to be noted that the anomaly is some 800 meters wider than the orebody, presumably because of a surrounding zone of disseminated pyrite.

Fig. 7 shows the response over an entirely different type of orebody, the *E* and *EL* orebodies at Lynn Lake, Manitoba. The scale of operations is reduced here: to discriminate those relatively narrow bodies, an electrode spacing of about 100 ft was used as opposed to 300 meters at Quellaveco, and readings were taken every 50 ft instead of every 100 meters. The smaller *E* body gives a better response than the *EL*. Some reasons for this are: 1) the *EL* body has massive sulfides, whereas the *E* is more disseminated,* and 2) the overburden is deeper over the *EL*. While both these bodies are adequately detected from their immediate surroundings, varying rock backgrounds reduce the certainty of the method in this area. For instance, not far to the west of the *EL* a quartzite formation gave response in the 50's, higher than that obtained over the *EL* itself. Disseminated pyrite possibly contributed to the high quartzite response.

* The overvoltage method works best with disseminated sulfides.

A contour map of anomalous overvoltage response provides a good picture of the distribution of sulfide mineralization; in regions where the depth to top of sulfides is less than about a third the electrode spacing and resistivity contrasts are not extreme. An example is given in Fig. 8, which is from a prospect in Peru; the contours here include a background response of about 5. Drilling in the highs provided approximate confirmation of the distribution in a limited portion.

A reading on one electrode spacing only gives no indication of depth of cover. This information can be obtained from expanders. An expander is a series of readings at different electrode spacings taken at one station. The results are then compared with type curves. In a great many cases the simple two-layer approximation is adequate. The derivation of two-layer type curves has been discussed under Type Curves. The investigator solves for depth and for anomalous response or mineralization factor of the underlying zone. The examples below are plotted linearly for greater clarity, but the method of solution requires the field results to be plotted on two-cycle logarithmic paper of the same size as the type curve paper. An expander is entirely analogous to the vertical profile of resistivity surveys.

Fig. 9 shows an expander taken at San Manuel, Ariz., plus a geological section in the region. The surrounding pyrite mineralization presumably renders the two-layer case applicable. This example is particularly interesting in illustrating how such deep mineralization as San Manuel's is detectable.

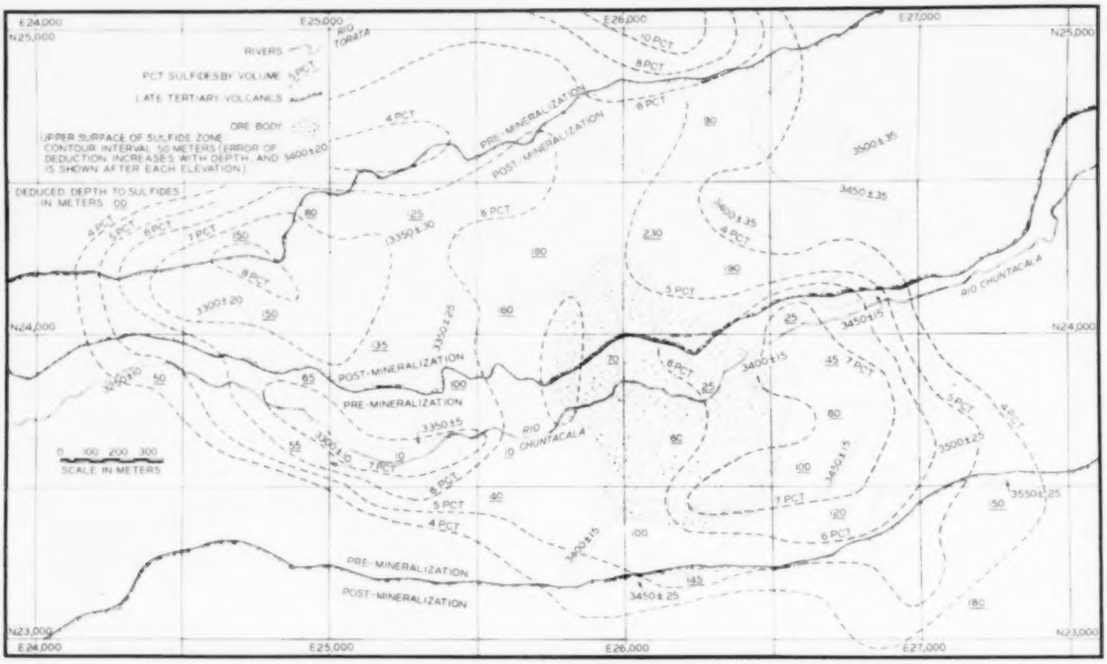


Fig. 11—The sulfide distribution at Cuajone, Peru, as deduced from the overvoltage data. Note the great variation in depth to the top of the sulfides. The mineralization that is outside the orebody consists mostly of pyrite.

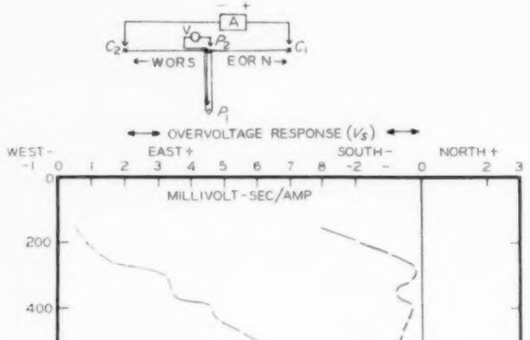


Fig. 12—Nababeep West, South Africa, borehole WP 12. The direction of mineralization from a drillhole is indicated here by the overvoltage azimuth survey.

An expander across the south end of the orebody at Cuajone, Peru (Fig. 10) gives depth to sulfides as 100 meters. Depth actually is about 90 meters.

With the aid of readings on more than one electrode spacing over a large area, it is possible to obtain mineralization factors and depths at a great number of points and then to contour this deduced data. At Cuajone two electrode spacings, one twice the other, were used on every line throughout the anomalous area, and additional control was provided by short spacing readings on several lines and by a few formal expanders. Fig. 11 shows a portion of the deduced mineralization and top of sulfide contour map; Fig. 11a, an aerial photograph of the region, illustrates to some extent the type of topography. For mineralization, it was assumed that a mineralization factor of 10 represented 1 pct sulfides by volume.* Depth to sulfides varies from less than 10 meters in the Chuntacala Valley to more than 160 meters where the late Tertiary volcanics cap the pampa or mesa to the north. The Cuajone orebody has now been extensively drilled and a rough outline is shown on the map. The deduced mineralization extends more than a kilometer to the west and more than half a kilometer to the east of the orebody, also (not shown here) far to the northwest. The deduced mineralization is at some points actually higher on the rim than directly over the orebody. The mineralization rim is disseminated pyrite. The drilling has in general verified the deduced mineralization pattern, but only relatively. A recent study of the assays from 35 drillholes has revealed that predicted sulfide content was on the average 1.95 times actual

*** This factor was based on tests made in Arizona.**

Fig. 11a—Air photo of Cuajone site shows steep hillsides, especially bordering the Rio Torata.

sulfide content. If this correction had been known in advance, the probable error of mineralization prediction at any point would have been about 30 pct of the predicted sulfide content, or less than 1 pct sulfides by volume. The probable error of depth prediction at Cuajone was 10 meters.

The overvoltage method has been tried in drillholes. This application, though it has given useful indications, has not had the widespread success that was first expected. One major problem has been correcting for the masking effect of low resistivity fluid in the drillhole, especially when working in very high resistivity Pre-Cambrian formations.

One important sideline to drillhole work is azimuth determinations. Once a significant anomaly is obtained in a drillhole using normal electrode arrays, direction is determined by placing the two current electrodes on surface an equal distance on each side of the collar, lowering one potential electrode down the hole, and measuring the overvoltage response with respect to a reference electrode. A positive response indicates that the source of the anomaly lies in the direction of the negative current electrode and vice versa. Two azimuth runs (north-south and east-west) are necessary to fully establish direction. Results in Nababeep West, South Africa, drillhole No. 12 (Fig. 12), suggest that in the upper part of the hole mineralization lies chiefly west, whereas in the lower part it lies chiefly to the south. These deductions were confirmed in the course of drilling the orebody.

There remain to be mentioned those unfortunate cases where overvoltage anomalies are not caused by sulfides.

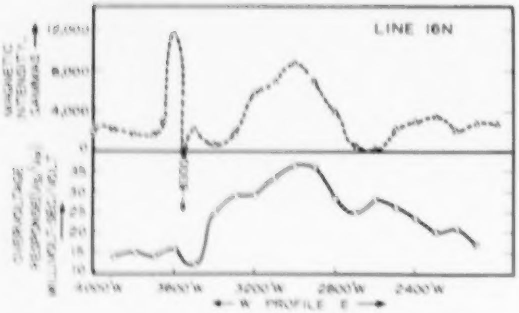


Fig. 13—Magnetic and overvoltage profiles at Engels, Calif. Overvoltage anomaly is attributed to magnetite.

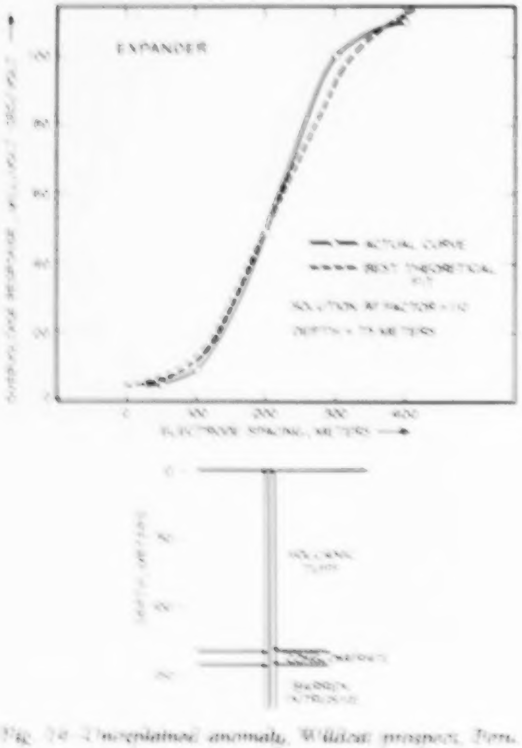


Fig. 14—Unexplained anomaly, Wildcat prospect, Peru.

Magnetite, being a metallic substance, gives an overvoltage response. An example of an anomaly presumably caused by disseminated magnetite comes from Engels, Calif. (Fig. 13). There is good correlation between the overvoltage and magnetic profiles. Of course the presence of an associated magnetic anomaly is not necessarily unfavorable. The two Lynn Lake examples both had excellent magnetic anomalies also.

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A wildcat anomaly obtained in Peru is still not satisfactorily explained. This occurred in a trough of post-mineral volcanic tuff. The expander taken at the center of the anomaly is shown in Fig. 14. Mineralization was predicted at less than 100 meters, the best solution being about 75 meters. In fact, drilling disclosed no lithological change for nearly twice this depth and the basement was only negligibly mineralized.

Victor Mayper[8] has shown that clay minerals with high ion exchange capacity can give a considerable overvoltage response. Notable extraneous anomalies were obtained in low resistivity phyllites in South West Africa and in certain schists in British Columbia.

The process of taking an overvoltage reading provides a resistivity reading automatically. The resistivity data are of direct use to the overvoltage survey in providing information necessary in depth calculations. A resistivity survey also has many well known applications—such as determining depth of overburden—and in itself is often a guide to mineralization. Porphyry coppers, for example, offer a fairly limited range of resistivity values. Most of the examples given in this article have accompanying resistivity anomalies. It is standard practice always to consider overvoltage results in conjunction with resistivity data.

Despite some unforeseen complications, e.g., the high response from certain nonsulfide material, the overvoltage method has proved its usefulness in detecting and outlining disseminated sulfide mineralization, even at depths as great as 200 meters.

The following firms have kindly granted permission to publish various items of information: Newmont Mining Corp., American Smelting & Refining Co., Cerro de Pasco Corp., San Manuel Copper Corp., Sherritt Gordon Mines Ltd., and O'okiep Copper Co. Ltd.

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* These items are private company papers, but it is hoped that they will soon be presented in a monograph to be published by the Pergamon Press.

Discussion of this article sent (2 copies) to AIME before Apr. 30, 1959, will be published in Mining Engineering.

PROCESSING CALIFORNIA BASTNASITE ORE

by CHARLES J. BAROCH, MORTON SMUTZ, and EDWIN H. OLSON

IN 1949 an orebody containing some 10 billion lb of recoverable rare earth metals was discovered in the Mountain Pass district of San Bernardino County, California.[1] The following year Molybdenum Corp. of America purchased a number of claims at the deposit and a mill on the property which had been constructed to process the district's gold ores.

Average composition of ore from the San Bernardino deposit is less than 0.1 pct ThO_2, 25 to 35 pct calcite, 10 pct rare earth oxides, 15 to 20 pct silica, and 30 to 40 pct barite.[2] Rare earth content of the ore analyzes 50.7 pct CeO_2, 4.2 pct Pr_6O_{11}, 11.7 pct Nd_2O_3, 1.3 pct Sm_2O_3, and 34.3 pct La_2O_3 and others.[2]

By floating the barite and depressing the bastnasite, Molybdenum Corp. of America produced a concentrate that contained 60 to 70 pct rare earth oxides. More than 25 pct of the rare earths, however, was lost in flotation. It seemed likely that a more economical method could be found.

At the present time most of the rare earths in the concentrates are converted to rare earth chlorides, which are either fed to ion exchange columns for separation of the individual rare earths or reduced to misch metal. Current investigation shows that solvent extraction may achieve this separation more economically than ion exchange. The study reported here was undertaken to develop an economical method of preparing a pure rare earth nitrate mixture from California bastnasite ore. Results indicated that more than 98 pct of the rare earths could be recovered.

Several possibilities were considered for processing bastnasite ore, but only the method outlined in Fig. 1 was investigated. This involved leaching the calcined ore with nitric acid. The nitric acid process is discussed under the following headings: 1) calcination of the ore, 2) leaching of calcine, 3) filtration studies, 4) solvent extraction, and 5) nitric acid recovery.

C. J. BAROCH, Junior Member AIME, formerly at the Ames Laboratory of the U. S. Atomic Energy Commission, Ames, Iowa, is now in the Army. M. SMUTZ is Assistant to the Director and E. H. OLSON is an Associate Engineer at the Ames Laboratory.
TP 4794L. Manuscript, May 21, 1958. AIME Trans., Vol. 214, 1959.

Calcination: Since California bastnasite ore contains 25 to 35 pct calcite and about 10 pct bastnasite, calcination was needed to reduce the violence of

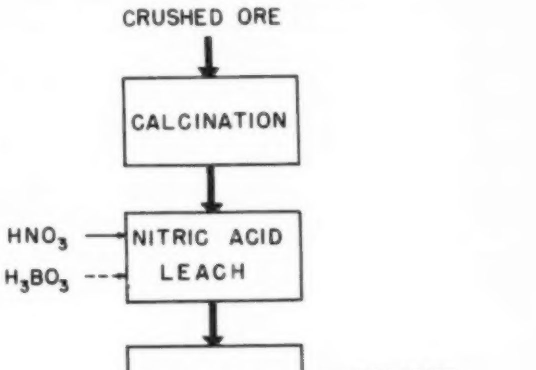


Fig. 1—Proposed method of processing bastnasite ore.

reaction with nitric acid. Weight losses of 26 to 30 pct were obtained by calcining the ore at 900°C for 1 hr or at 800°C for 4 hr. Longer periods resulted in no further weight loss.

Leaching the Calcine: When the bastnasite calcine was leached with 30 pct nitric acid for 8 to 10 days, rare earth recoveries of only 45 to 50 pct were

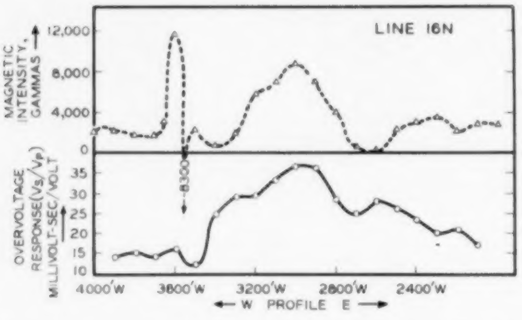
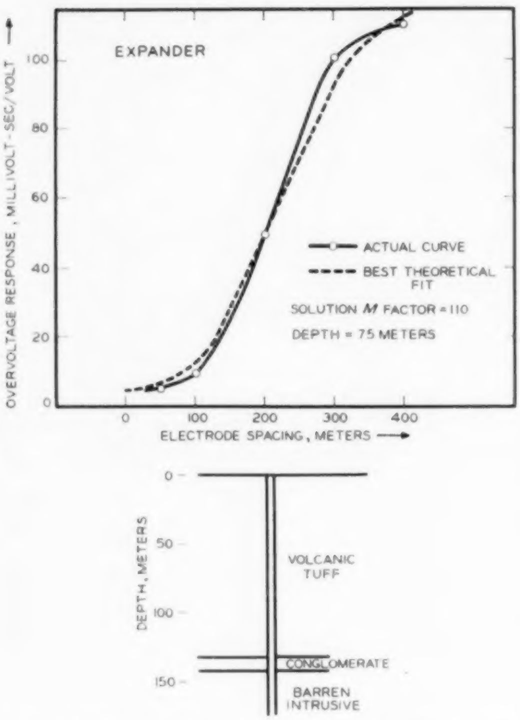


Fig. 13–Magnetic and overvoltage profiles at Engels, Calif. Overvoltage anomaly is attributed to magnetite.



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* These items are private company papers, but it is hoped that they will soon be presented in a monograph to be published by the Pergamon Press.

Discussion of this article sent (2 copies) to AIME before April 30, 1959, will be published in MINING ENGINEERING.

PROCESSING CALIFORNIA BASTNASITE ORE

by CHARLES J. BAROCH, MORTON SMUTZ, and EDWIN H. OLSON

IN 1949 an orebody containing some 10 billion lb of recoverable rare earth metals was discovered in the Mountain Pass district of San Bernardino County, California.[1] The following year Molybdenum Corp. of America purchased a number of claims at the deposit and a mill on the property which had been constructed to process the district's gold ores.

Average composition of ore from the San Bernardino deposit is less than 0.1 pct ThO_2, 25 to 35 pct calcite, 10 pct rare earth oxides, 15 to 20 pct silica, and 30 to 40 pct barite.[2] Rare earth content of the ore analyzes 50.7 pct CeO_2, 4.2 pct Pr_6O_{11}, 11.7 pct Nd_2O_3, 1.3 pct Sm_2O_3, and 34.3 pct La_2O_3 and others.[2]

By floating the barite and depressing the bastnasite, Molybdenum Corp. of America produced a concentrate that contained 60 to 70 pct rare earth oxides. More than 25 pct of the rare earths, however, was lost in flotation. It seemed likely that a more economical method could be found.

At the present time most of the rare earths in the concentrates are converted to rare earth chlorides, which are either fed to ion exchange columns for separation of the individual rare earths or reduced to misch metal. Current investigation shows that solvent extraction may achieve this separation more economically than ion exchange. The study reported here was undertaken to develop an economical method of preparing a pure rare earth nitrate mixture from California bastnasite ore. Results indicated that more than 98 pct of the rare earths could be recovered.

Several possibilities were considered for processing bastnasite ore, but only the method outlined in Fig. 1 was investigated. This involved leaching the calcined ore with nitric acid. The nitric acid process is discussed under the following headings: 1) calcination of the ore, 2) leaching of calcine, 3) filtration studies, 4) solvent extraction, and 5) nitric acid recovery.

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TP 4794L. Manuscript, May 21, 1958. AIME Trans., Vol. 214, 1959.

Calcination: Since California bastnasite ore contains 25 to 35 pct calcite and about 10 pct bastnasite, calcination was needed to reduce the violence of

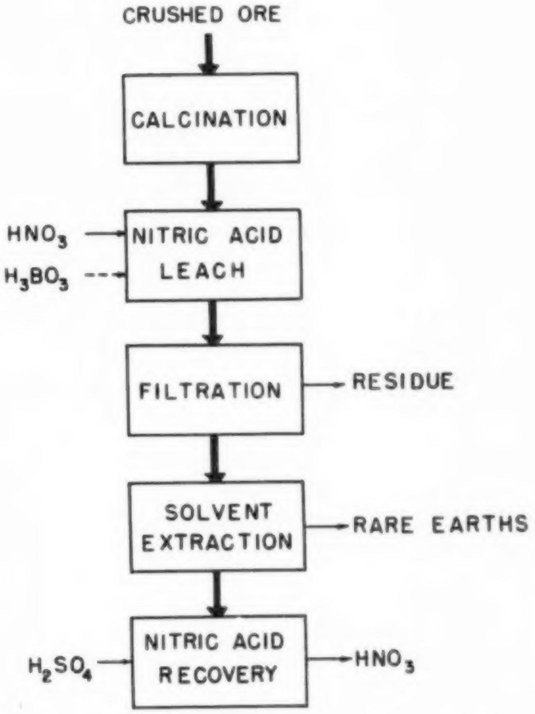

Fig. 1—Proposed method of processing bastnasite ore.

reaction with nitric acid. Weight losses of 26 to 30 pct were obtained by calcining the ore at 900°C for 1 hr or at 800°C for 4 hr. Longer periods resulted in no further weight loss.

Leaching the Calcine: When the bastnasite calcine was leached with 30 pct nitric acid for 8 to 10 days, rare earth recoveries of only 45 to 50 pct were

obtained. Because of the low rare earth recoveries, another method of opening the ore was sought.

C. J. Rodden[3] had stated that the rare earth fluorides were soluble in a mixture of concentrated nitric acid and boric acid. Leaching the calcine with this mixture for 8 to 10 days indicated rare earth recoveries of more than 90 pct. Further tests showed that boric acid was not required. The major disadvantage was the high cost of the concentrated nitric acid, but the commercial preparation of about 57 pct, costing only 2.2¢ per lb, proved equally effective.

Additional tests showed that leaching the calcine for only 1 hr liberated over 93 pct of the rare earths. The percentage recovered was not appreciably affected by increasing the leach time to 2 hr or more. These experiments also revealed that size of calcine particles had very little effect on the percentage of rare earths recovered. For economic reasons, −10 mesh particles were considered satisfactory.

A 1-hr leach and a nitric acid/calcine weight ratio of 2.2 were required to liberate 93 pct of the rare earths. Larger-scale leachings using the same acid/calcine ratio resulted in the liberation of over 99 pct of the rare earths from the calcine. This minimum nitric acid/calcine weight ratio was not the optimum required for the overall process. It will be shown that efficiency of filtration and of solvent extraction depends on the ratio of acid to calcine weight.

Filtration Studies: It is difficult to predict accurately, from small-scale laboratory tests, the filter area needed in commercial plants. Several standard tests have been devised, however, from which it is possible to predict semi-quantitatively the requirements for a large plant.

In the filtration experiments Eimco media No. Dy-452 was used on a standard leaf test. This cloth resisted the attack of hot concentrated nitric acid and was capable of retaining particles as fine as 200 mesh. A vacuum of 10 in. of mercury was used, and a plot was made of time vs volume of filtrate.

A standard lot of calcined ore was prepared for the tests. Owing to the small size of the leaching vessel, two equal batches of slurry were prepared and mixed, and when the temperature of the combined slurries dropped to 45°C, the filtration tests were started. Results showed that filtration was best with an acid/calcine weight ratio of 2.96. To simplify calculations, a ratio of 3.00 was used.

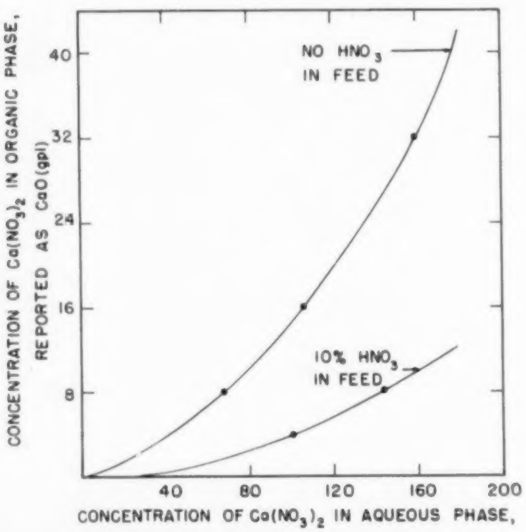

Fig. 2—The calcium nitrate distribution curves used in determining amount of calcium transferred to TBP.

Table I. Results of Typical Solvent Extraction Operation

Stage	Volume, Cm³		Analysis of Exit Streams	
	In	Out	RE_2O_3, Gpl	CaO, Gpl
Extract 1				
Aqueous	716*	690	25.74	91.10
Organic	358	400	32.26	1.23
Extract 2				
Aqueous	690	675	12.34	91.86
Organic	345	360	26.51	3.71
Extract 3				
Aqueous	675	658	2.58	87.58
Organic	338	340	19.31	9.00
Extract 4				
Aqueous	658	655	0.00	79.56
Organic	329	330	4.85	16.07
Scrub 1				
Aqueous	96	93	20.16	76.46
Scrub 2				
Aqueous	96	96	40.12	28.58
Organic	1430	1430	17.38	0.00

* The feed to Extract 1 of the extraction section consisted of scrub 1 and 2, 25 cm³ of 57 pct HNO_3, and 500 cm³ of fresh feed (see Fig. 4). The fresh feed contained 53.95 gpl RE_2O_3 and 107.40 gpl CaO and had an acid normality of 2.46.

Table II. Composition of Various Streams of a Bench-Scale Test

Material	Quantity	RE_2O_3, G	CaO, G	SiO_2, G	$BaSO_4$, G	HNO_3, G
Raw ore	7044.2 g	734.4	1554.3	118.3	1767.4	—
Calcine	5000.0 g	733.1	1538.1	91.5	1690.0	—
Leached ore	1876.0 g	—	—	92.3	1667.0	—
HNO_3 feed	11.0 L	—	—	—	—	8299.7
Leach liquor	14.0 L	755.5	1503.6	—	—	2173.2
Additional HNO_3 to feed	0.7 L	—	—	—	—	528.8
Raffinate	18.1 L	—	1440.0	—	—	120.9
TBP product	40.0 L	699.9	—	—	—	2424.0
$CaSO_4$ precipitate	3190.0 g	—	1488.0*	—	—	—
HNO_3 recovered	6.3 L	—	—	—	—	4752.0

* Estimated from weight of the $CaSO_4$ precipitate.

Filterability was influenced by many properties of the slurries, such as viscosity of the clear leach liquor and size of the reacted calcine particles. When the calcine was leached with a nitric acid/calcine weight ratio of 3.00 or less, a nearly saturated solution of calcium nitrate was obtained. When the solution cooled to room temperature, some calcium nitrate and some of the rare earths were crystallized. Crystallization of the calcium nitrate greatly reduced filterability of the slurries.

Crystallization of the calcium nitrates and rare earth nitrates can be prevented by diluting the leach liquor with water or by heating the liquor to 40°C or higher. It would be much more costly to maintain the leach liquor at 40°C or higher than it would be to dilute it. Most of the calcium nitrate was kept in solution, therefore, by adding water equal to 35 to 40 pct of the volume of the leach liquor.

Still another variable influenced filtration characteristics of the slurries—the greater the tetravalent cerium concentration, the poorer the filterability. Addition of hydrogen peroxide, powdered iron, or magnesium to the slurry reduced the tetravalent

cerium concentration, but the cost of these reagents would not be offset by the reduction in filtration costs.

No effort was made to determine the minimum amount of water required to wash the filter cake, but over 99 pct of the soluble rare earths was recovered when the cake was washed with water equivalent to about 25 pct of the volume of the leach liquor. It can be assumed, therefore, that the amount of water necessary to keep all of the calcium nitrate in solution would be a good measure of the amount required to wash the filter cake satisfactorily.

Solvent Extraction: Solvent extraction should be one of the cheapest and simplest methods of removing the combined rare earth nitrates from the leach liquor. According to Knapp,[4] the distribution coefficient of the combined rare earths was greatly increased by the presence of salting agents such as calcium and ferric nitrates. Since California bastnasite ore contains about twice as much soluble calcium as rare earths, the distribution coefficient of the combined rare earth nitrates should be very favorable. The distribution curves for the combined rare earth nitrates were obtained by three to six successive contacts of the leach liquor with tributyl phosphate (TBP) which contained no diluent. The TBP used in these tests was pre-equilibrated with nitric acid of varying strengths so that a series of distribution curves could be obtained. The equilibrium curves indicated that in most cases the distribution coefficient of the combined rare earths varied inversely as the nitric acid concentration in the TBP. In determining the rare earth distribution curves, it was noticed that some calcium was transferred to the TBP. However, when the nitric acid concentration in the fresh TBP was greater than 1.26 *N*, no calcium was detected in the TBP product.

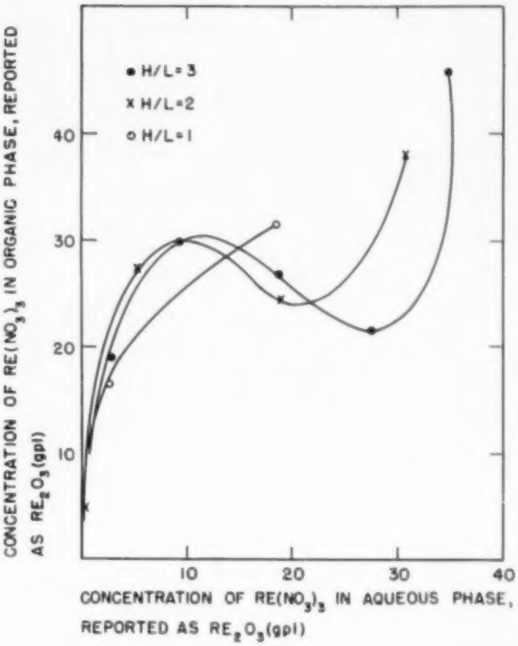

Fig. 3—Effect of H/L volume ratio on total rare earth nitrate distribution curve. Best cocurrent extraction is obtained with H/L ratio of 2, using four extract stages, but for cocurrent extraction to apply in this process, the calcium must be separated from the rare earths.

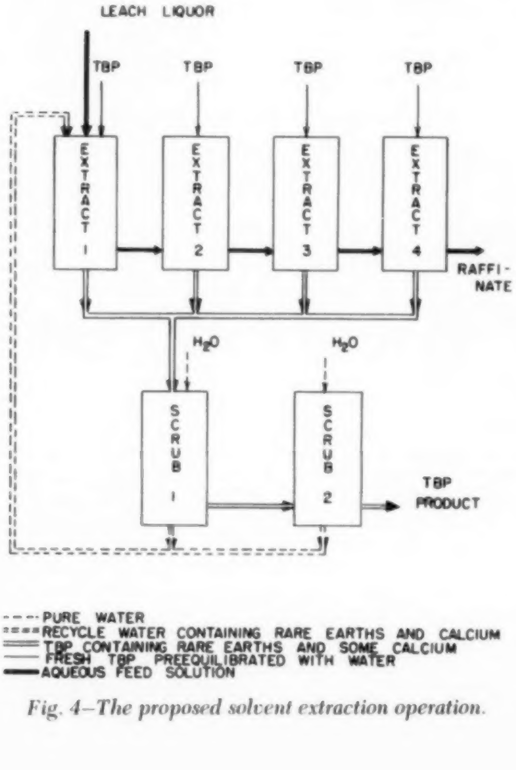

Fig. 4—The proposed solvent extraction operation.

To determine the amount of calcium transferred to the TBP, the distribution curve for a calcium nitrate solution was determined. The calcium nitrate distribution curves in Fig. 2 indicate that the distribution coefficient of the calcium nitrate varies inversely as the nitric acid concentration.

Effectiveness of the solvent extraction operation was determined by making a simulated countercurrent solvent extraction run. The rare earth distribution curve for this run did not duplicate the curves obtained using cocurrent extractions. In fact, in some of the extract stages the rare earth concentration in the aqueous phase increased about 50 pct. Also the extraction was less than 35 pct effective in removing rare earths from the aqueous feed.

Several attempts were made to improve the efficiency of countercurrent extractions. However, varying the number of extract stages, the aqueous to organic (H/L) volume ratio, the nitric acid concentration in the TBP, and the amount of scrub solution had very little effect. Varying the operating conditions seemed only to influence the distribution curve, and in no instance was the rare earth recovery greater than 50 pct.

Cocurrent extraction did overcome the difficulties found in the countercurrent operation, but the application of cocurrent extraction might be limited by the amount of calcium transferred to the TBP, as well as by the quantity of TBP required to recover 98 pct of the rare earths in the aqueous feed solution.

Effectiveness of the cocurrent extractions was determined by making several runs using H/L ratios of 1, 2, and 3. In all instances the TBP contained no diluent or nitric acid. The rare earth distribution curves obtained in the cocurrent runs

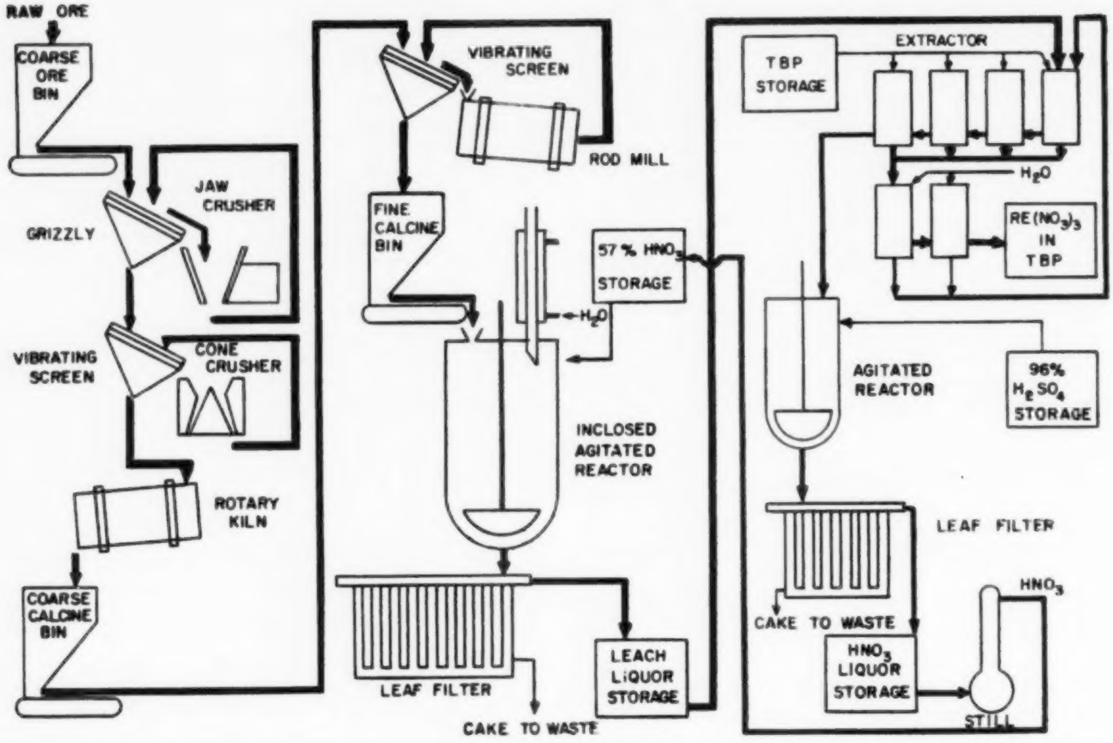

Fig. 5—Process or preparing rare earth nitrate mixture from California bastnasite ore gives over 98 pct recovery.

are shown in Fig. 3. Variability in the curves can probably be attributed to the difference in extractability of the tetravalent cerium, nitric acid, and the remainder of the rare earths.

Calculations using the curves in Fig. 3 indicated that best cocurrent extraction can be obtained with an H/L ratio of 2, using four extract stages. However, in order for the cocurrent extraction to be applicable to this process, the calcium must be separated from the rare earths. The low distribution coefficient of the calcium nitrate suggested that if the TBP containing both calcium and rare earth nitrates were scrubbed with water, the calcium nitrate might be preferentially extracted. The scrubbing operation proved to be effective—in fact, practically all of the calcium was scrubbed from the TBP by two cocurrent contacts of the combined TBP products with water, using an H/L ratio of 0.067 in each contact. Since some of the rare earths were also removed, the scrub solutions must be processed to recover them.

A method for recovering the rare earths in the scrub solution is outlined in Fig. 4. Several simulated runs with this method indicated that recycling the scrub solution had very little effect on the rare earth distribution curve and that a TBP product could be obtained which was virtually free of calcium.

Nitric Acid Recovery: In the leaching operation over 60 pct of the nitric acid is consumed by the calcite and other impurities. Therefore the nitric acid would have to be recovered from the calcium nitrate and other nitrate compounds. The nitric acid recovery operation would involve a chemical reaction between calcium nitrate and sulfuric acid which would produce calcium sulfate and nitric acid. The resulting slurry would be filtered and the filtrate processed by extractive distillation.

Details of the extractive distillation operation were not investigated in this project. A 96 pct sulfuric acid to raffinate volume ratio of 0.15 would be required to liberate most of the nitric acid. Because of the variable composition of the raffinate, however, an acid to raffinate volume ratio of 0.18 might prove more satisfactory.

BENCH-SCALE INVESTIGATIONS

Several bench tests of the process outlined in Fig. 5 provided information for a more reliable material balance and also indicated where difficulties might arise in scaling up the process. By increasing the size of the operation, it was possible to study the processing steps more easily.

About 7200 g of raw ore were thoroughly mixed, sampled, and calcined. After the calcine had cooled, it was crushed to pass a 10-mesh screen, remixed, and sampled. Five kilograms of the calcine were slowly fed to 11,000 ml of about 57 pct nitric acid. The slurry was thoroughly agitated throughout the leaching operation. Originally it was planned to feed the calcine at a rate of 5000 g per hr, but its reaction with nitric acid was so violent that excessive amounts of steam and nitric acid were produced. To minimize the loss of nitric acid, the calcine feed rate was reduced to about 4000 g per hr. The reaction slurry was then agitated for one additional hour after all the calcine had been fed.

The nitric acid vapors evolved by the hot slurry made the filtration immediately after leaching very difficult; for this reason the slurry was allowed to cool for about 2 hr before it was filtered. The filter cake was then washed with about 2500 ml of water.

Any calcium nitrate that crystallized upon cooling was dissolved by diluting the leach liquor with water. A small portion of the leach liquor was then used in the solvent extraction operation.

In the first experiment with cocurrent solvent extraction, the feed solution had an acid normality of about 2.5, and the run yielded a TBP product that contained about 1.3 gpl of calcium oxide. Therefore, a second solvent extraction run was performed in which the nitric acid concentration of the fresh feed was increased to about 3.0*N*. The use of this additional nitric acid yielded a TBP product that was free of calcium. Results of this solvent extraction run are presented in Table I.

A fraction of the raffinate from the solvent extraction operation was processed to recover the nitric acid that had combined with the calcium and other impurities. Over 92 pct of the nitric acid in the raffinate was recovered in this operation. Undoubtedly greater nitric acid recoveries can be obtained; however, the optimum conditions could be more easily determined in a pilot plant operation.

The various streams, raw materials, and products of the bench-scale tests were analyzed for rare earths, calcium, barium, silica, and nitric acid content. Table II presents the analysis of the various streams of a typical bench-scale operation.

Summary: The process developed during this investigation consisted of the following steps:

1) Calcination of ore for 1 hr at 900°C or 4 hr at 800°C.

2) Digestion of the calcine in 57 pct nitric acid for 1 hr. Almost all of the rare earths and calcium were readily dissolved from −10 mesh calcine particles. An acid to calcine weight ratio of 3.00 was required.

3) Filtration of reaction slurry and washing of filter cake. At least a 35 pct dilution of the leach liquor was required to keep all of the calcium nitrate in solution.

4) Recovery of the combined rare earth nitrates from the leach liquor in a solvent extraction operation using TBP as the solvent. The operation was performed in four cocurrent extract stages using an H/L volume ratio of 2.00 in each stage. To remove the calcium nitrate, the TBP products were combined and scrubbed twice with water, using an H/L ratio of 0.067. The scrub solutions were recycled to the first extract stage. An overall H/L ratio of 0.38 was required in the solvent extraction operation.

5) Recovery of the nitric acid combined with the calcium and other impurities. A sulfuric acid to raffinate volume ratio of 0.18 was required to liberate most of the nitric acid. Nitric acid in the reaction slurry was recovered by filtration and then concentrated by extractive distillation.

Results obtained in this study showed that over 98 pct of the rare earths could be recovered. About 18 pct of the nitric acid fed to the leaching operation was combined with the rare earths or was lost in the nitric acid recovery operation.

This article is a condensation of a dissertation presented by C. J. Baroch in partial fulfillment of the requirements of a Doctor of Philosophy degree at Iowa State College, Ames, Iowa. Work was performed in the Ames Laboratory of the U. S. Atomic Energy Commission.

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INTERACTION OF MINERALS WITH GASES AND REAGENTS IN FLOTATION

by IGOR PLAKSIN

Interaction of sulfide minerals and native metals with reagents in flotation is largely determined by particle-surface changes resulting from action of the medium and dissolved gases.

A number of early studies were devoted to the selective action of various gases on minerals.[1] McLachlan[2] paid attention to oxygen as the most active gas affecting flotation properties of sulfide minerals. Opposing views were expressed concerning the influence of sulfide oxidation on xanthate flotation: Taggart[3] recognized the positive influence of oxygen, and I. W. Wark[4] rejected it. Shvedov[5] offered the hypothesis of partial oxidation. As to the influence of such gases as hydrogen[6] and carbon dioxide on mineral floatability, opinions also differed.

Effect of Oxygen on Sulfide Mineral Flotation: Research on the influence of oxygen and other gases on the floatability of sulfide minerals and native metals has been carried out at the Institute of Mining of the USSR Academy of Sciences in

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Behavior of both sulfide and nonsulfide minerals depends on the concentration of oxygen in solution. The response of these minerals to variations in oxygen level is affected significantly by their crystal structure.

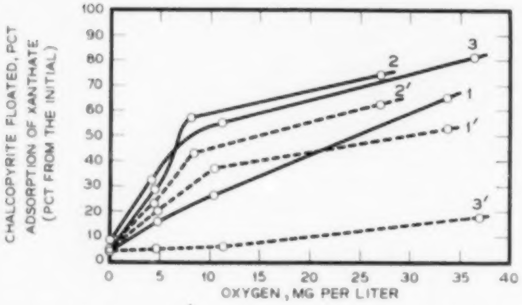

Fig. 1—Influence of oxygen on chalcopyrite flotation (from mixture with quartz) in an argon atmosphere, following additions of S-35 marked potassium ethyl xanthate. Solid lines represent mineral recovery and dotted lines the xanthate adsorption. Curves 1 and 1', xanthate addition at 10 g per ton; curves 2 and 2', xanthate addition at 100 g per ton; curves 3 and 3', xanthate addition at 1000 g per ton; and terpineol, 50 g per ton.

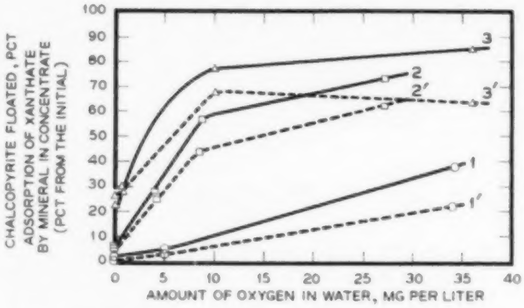

Fig. 2—Influence of minor quantities of oxygen on chalcopyrite flotation (from mixture with quartz) in atmospheres of hydrogen, argon, and carbon dioxide, using potassium ethyl xanthate (100 g per ton) and terpineol (50 g per ton) as reagents. The solid lines represent mineral recovery and the dotted lines xanthate adsorption. Curves 1 and 1', with hydrogen as the principal gas; curves 2 and 2', with argon as the principal gas; curves 3 and 3', with carbon dioxide as the principal gas.

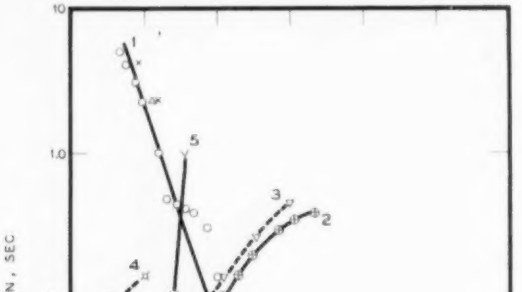

Fig. 3—Glembotsky times of bubble adhesion as a function of duration of mineral-water contact for pyrite, galena, and chalcopyrite. Curve 1, chalcopyrite (various experiments); curve 2, galena ground in water; curve 3, galena ground in air; curve 4, oxidized galena that is preliminarily sulfidized; curve 5, pyrite.

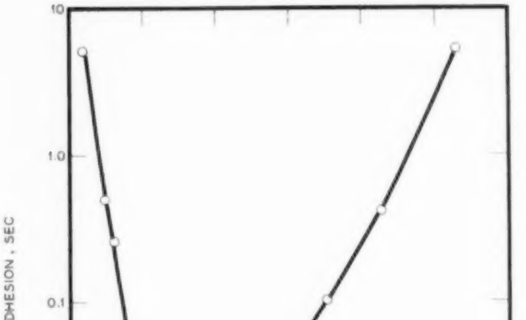

Fig. 4—Glembotsky times of bubble adhesion to the galena as a function of the mineral-water contact, the water being at first deoxygenated and then oxygenated.

Moscow.[7-15] The basic method was flotation of ground minerals in water containing strictly measured amounts of oxygen in an inert gas atmosphere. A standard mineral surface was achieved by grinding and classifying pure sulfide mineral in an oxygen-free atmosphere (nitrogen or argon). Thus treated, the mineral was floated in deoxygenated water by an inert gas, or in water with a definite oxygen content (from traces to 20 to 30 mg per liter).

The apparatus was considerably more elaborate than usual in flotation testing. It included a device for purifying the gas used and for distilling water under a controlled gas atmosphere. Special grinding mills, screens, and flotation cells were designed to permit carrying out these operations under closely controlled gas composition.

Floatability increases with increased oxygen content, in different ratios for various minerals. The most complete flotation of galena is obtained with water having an oxygen content of 1 to 1.5 mg per liter, whereas complete flotation of chalcopyrite requires tens of milligrams of oxygen per liter. Different quantities of oxygen in solution are necessary for the flotation of various sulfides, and rate of floatability increases for various sulfides depending on their activity with relation to oxygen. This has been established by the flotation of monomineral powders as well as by flotation of mineral mixtures. The following mineral sequence has been found with regard to increasing oxygen content of the atmosphere in order to attain complete flotation with a collector: galena, pyrite, sphalerite, chalcopyrite, pyrrhotite, arsenopyrite.

The nonfloatability of chalcopyrite in the absence of oxygen was proved by experiments in the above-mentioned apparatus, using xanthates up to 1 kg per ton.[11] Fig. 1 shows the recovery of chalcopyrite obtained with various quantities of collector added at various oxygen levels; this recovery increases with oxygen level and is very low when the level is under 1 mg per liter. Fig. 1 also shows the xanthate consumed by the mineral (dotted lines) as determined by assays based on S-35.

Carbon dioxide favorably influences floatability of chalcopyrite. In this case, in the absence of oxygen, a noticeable recovery is obtained (Fig. 2). This action may be explained by the decrease of pH from about 7 to 6 in the presence of carbon dioxide; if pH is increased to 6 to 8 by addition of NaOH, this flotation effect practically disappears. Hydrogen does not increase the flotation of chalcopyrite;[18] in fact, it lowers it somewhat in comparison with argon or nitrogen, whose bubbles serve only as *carriers* for the mineral particles (Fig. 2).

It is interesting that conditions of gas composition producing good mineral recovery also produce substantial collector adsorption; conversely, lack of recovery corresponds to lack of adsorption. This is shown by the accord between solid and dotted lines in Figs. 1 and 2.

During flotation tests without collector or frother, floatability was noticed in the presence of oxygen. This natural floatability, or ability of minerals to float in the water without reagents, agrees with the oxygen sulfide activity sequence. In the absence of oxygen and reagents, however, none of the sulfides float. Experiments have proved that on the fresh surface of a sulfide mineral the oxygen is adsorbed first, and then the xanthate is fixed.

Action of oxygen and other gases upon sulfide minerals and native metals has been studied by several methods other than direct flotation: measurement of the contact angle, measurement of adhesion time of particles to bubbles in an electronic contact device,[16] determination of oxygen abstraction from a sulfide mineral pulp,[17] study of the ionic composition of the liquid phase of sulfide suspensions, measurements* of irreversible sulfide-mineral potentials;[14,15] and others.[24,25] The data obtained make it possible to offer a consistent explanation of the phenomena observed. The study of the adhesion time of the mineral particles to air bubbles in the Glembotsky contact device[16] is particularly revealing. The Glembotsky apparatus is an electronic mechanism that evaluates the time required for adhesion to occur between a bubble and a mineral surface. In many measurements this time ranges from 10 to 10^{-3} sec.* Of course, the more hydrophobic the mineral, the shorter this induction time, and a completely hydrophilic surface should require an infinite induction time. Practically, if the induction time exceeds 5 m-sec the mineral is unfloatable. Measurements by the Glembotsky device using pure water on minerals prepared in various ways are summarized in Fig. 3, which illustrates not only the reproducibility of the data, but also substantial differences in behavior between the various sulfides. Also, galena (curves 2 and 3) behaves in much the same way whether prepared by grinding in water or in air, whereas oxidized galena preliminarily sulfidized behaves differently (curve 4).

The author's interpretation of these facts is that the change of surface properties of galena and pyrite proceeds rapidly, whereas for chalcopyrite it occurs slowly. Oxygen adsorption on galena is so rapid that in water with ordinary oxygen content (7 to 8 mg per liter), initial surface wettability is not detected. Galena wettability is found if the ad-

* We study the electrochemical unevenness of a surface of sulfide minerals,[24, 25] and also the electrochemical properties of mineral surface in general.[14, 15] We use a method of anode and cathode polarization of sulfide. We have studied by this method anode and cathode parts on the surface of a sulfide mineral. Uneven surface chemical compounds of sulfide minerals have been proved by these tests. Such unevenness of electrochemical potential on the sulfide mineral surface influences the distribution of reagents on mineral particles of flotation pulp.[26]

* Sven Nilson and then M. A. Eigeless were the first to register the time of contact between mineral particle and air bubble. Glembotsky's device is unique in that it is an electronic mechanism with a moving holder of the bubble.

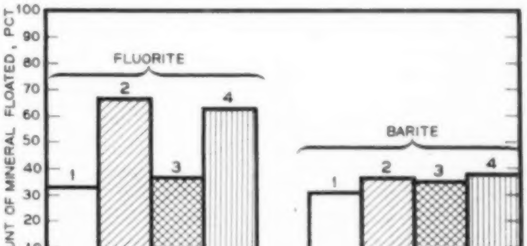

Fig. 5—Effect of aerating gas on recovery of fluorite (left) and barite (right). The numbers 1, 2, 3, and 4 refer to flotation condition as follows: 1, without deliberately added gases; 2, following oxygenation; 3, following nitrogen addition after oxygenation; 4, after second oxygenation. Numbers at base of blocks represent oxygen concentration in milligrams per liter.

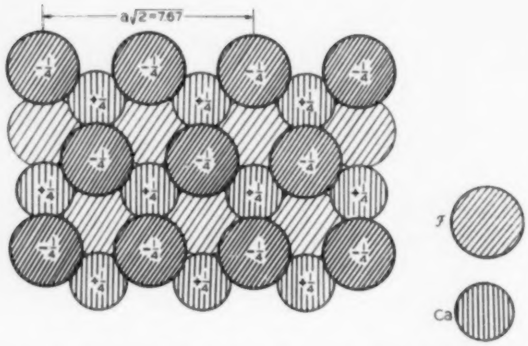

Fig. 6—Diagram illustrating the projection of the fluorite structure upon the cleavage plane (111).

hesion time measurements are conducted first in deoxidized water and then while oxygen is dissolved in the water (Fig. 4, left branch of the curve). These tests prove that sulfide mineral particles can adhere to a bubble in the presence of oxygen.[18] It should be noted that in this study the relations obtained between galena adsorption of oxygen dissolved in water and galena adsorption of oxygen from the air are very close to the results published in Hagihara's work on electron diffraction of galena oxidation.[19]

Thus it has been established that in the absence of oxygen the fresh surface of a sulfide mineral or of a metal is wettable to some extent. The oxygen dissolved in water is adsorbed on the surface of minerals before the other gases (nitrogen, carbon dioxide, etc.) are adsorbed. Plaksin and Bessonov[9] distinguish successive stages of the action of oxygen with a transition from a labile form of fixing through a stable one to forming of chemical compounds, which can be shown, for instance, by the following sequence: 1) adsorption, 2) activated adsorption with fixing of oxygen, 3) oxidation of the surface with oxygen diffusion in the surface layer. For various sulfides and metals these stages occur at different rates, depending on the chemical activity of the sulfide (or metal) for oxygen. Flotation properties of mineral surfaces under the influence of oxygen and collector change accordingly. In the initial period of its action, oxygen promotes dehydration of the mineral surface, facilitating penetration of xanthate groups and their consequent fixation. In case of long exposure to oxygen the flotation properties of a surface become worse.

These observations and results have made it possible to classify gold ores and the sulfides of nonferrous metals as to their oxygen demand during flotation. Proceeding from the experience of one USSR dressing plant, a direct selective method of flotation for copper-lead-zinc ores with oxygen control has been introduced.

Experiments have been made with controlled oxygen content in flotation of sulfide minerals with xanthate in respect to modifiers, activators, and depressants, in particular on the action of sodium sulfide.[20] The experiments show that a certain average concentration of sodium sulfide exists when adsorption of the collector by galena reaches a peak. The peak is explained by an initial oxidation of the galena and agrees well with the depression of galena due to excess sodium sulfide.[21]

Effect of Oxygen on Flotation of Nonsulfide Minerals: The influence of oxygen and other gases on the floatability of nonsulfide minerals (fluorite, barite, calcite, quartz) has also been studied, with unexpected results. Phosphorite was studied first. It was found that flotation properties of the ore changed in relation to the gas blown into the pulp. The phosphorite ore was ground to 0.15 to 0.05 mm and after preliminary desliming was placed in a laboratory flotation machine of 350-cc capacity. The machine had a gas-tight cover and at the side there was a device for supplying gas to the flotation chamber. Reagents used were carboxylic acid (or tall oil soap), caustic soda, and sodium silicate. The other experiments were carried out first without addition of gases. When gases were used they were allowed to remain in contact with the pulps for 3 to 75 min. After gasification, the flotation reagents were introduced and the pulp and reagents thoroughly mixed, after which flotation was carried out. Temperature of the pulps was maintained at 18° to 20°C for 5 to 60 min. In the experiments with oxygen and air, flotation was carried out with tap water and distilled water, but for experiments with nitrogen fresh doubly-distilled water was used. It was found that introduction of air and/or oxygen in the pulps increased the recoveries, whereas nitrogen decreased them.[22-25]

Particles of fluorite and quartz (0.15 to 0.06 mm) were floated with sodium oleate (100 to 500 g per ton) by passing oxygen or nitrogen through the pulp for varying lengths of time. On addition of 100 g of sodium oleate per ton of fluorite the collector abstraction corresponded initially to 94 pct of a monomolecular layer and increased to 137 pct after 60 min. The thickness of the layer was approximately twice as great at 200 g per ton. Further increases in reagent addition are generally without effect. The change in thickness of the adsorbed layer brought about by continued oxygen treatment has no significant effect on recovery of fluorite or of quartz in the concentrate.[26]

The experiments, conducted under the usual precise conditions of flotation, showed that recovery of nonsulfide minerals in the froth product was greatly increased if oxygen had previously been admitted to the mineral surface.

Motion pictures made it possible to observe the progress of mineralization of the air bubble with mineral grains under dynamic conditions. The experiments were carried out in a cuvette with plane parallel walls, filled with water or with aqueous solution of the collector, in which the motion of the air bubbles and mineral particles was effected. Pictures were taken at 400, 900, and 1500 frames per sec with bubbles 2 mm in diam ascending at 18 to 27 cm per sec.

By speed filming, then, it was possible to determine that additional adsorption of oxygen on the surface of fluorite produces:

1) Higher mineralization of the bubble with the particles of fluorite.

2) Five to ten times longer contact between mineral grain and air bubbles.

3) Air flocculant attachment of fluorite grains to the air bubble.

Relation Between Crystal Structure and Influence of Gases: Study of the combined action of gases and

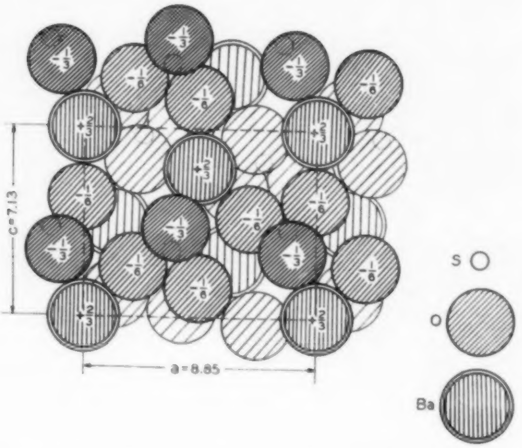

Fig. 7—Projection of barite structure upon cleavage plane (110). Here and in Fig. 6 the varying intensities of shading indicate distances of the ions from a geometrical plane that is tangent to the outermost atoms.

reagents on flotation of nonsulfide minerals brings out peculiarities that can be rationalized in terms of crystal structure. The complex effects for fluorite and barite are shown in Fig. 5. Blocks 1 and 2, left, show that oxygen markedly favors recovery of fluorite, while the corresponding blocks in the right half of the illustration show that there is no such effect with barite. Blowing nitrogen after oxygen (block 3) shows reversion to unoxygenated recoveries, while secondary oxygen blowing (block 4) restores the oxygen effect. Thus changes in the floatability of certain nonsulfide minerals by dissolved gases is not a general phenomenon.

To explain the flotation behavior of fluorite and barite, peculiarities of their crystal structure may be used, considering that the breaking of minerals occurs mainly along the cleavage faces and that the molecular-physical properties of the surfaces are determined by their crystal chemistry.

Study of the projection of fluorite and barite structures on the cleavage plane (Figs. 6 and 7) shows that the fluorite surface is distinguished by a low relief, and its surface molecular field is characterized by evenness and a small value for the intensity (the value of separate noncompensated electrostatic charges is not above ¼). On the contrary, the surface of barite is distinguished by substantial differences in the values of noncompensated charges and a radically expressed unevenness of their distribution. This, together with a certain softness in the structural setting of ions and the presence of great unevenness, predetermines the heterogeneity of the surface molecular field and its greater intensity in the places of localization—in the active centers, that is—having charges of ½ to 2/3.

Thus the wettability of a fluorite surface, which is low compared with that of a barite surface, is predetermined first by a lower value of the noncompensated charges, second by a poorer distribution of spaces for the adsorbed molecules of water. A part of the water molecules cannot approach the attracting ion up to immediate contact. The relative evenness of the surface, together with the absence of strong charges, promotes the instability and lability of the hydrate layer.

In a similar way, the peculiarities of the surface molecular field determine the behavior of gases dissolved in the pulp. As the result of screening by the adsorbed molecules of the surface force field, the total surface wetting is changed. In turn the diffusion of gases to the surface, which precedes the adsorption, depends on correct orientation of the dipoles of the hydrate layer. There is an apparent and direct relation between the degrees of surface wetting and the surface adsorption of gases.

The degree of hydrophobicity, which is increased as a result of the physical adsorption of gases from the solution, may on the whole be associated with the intensity value of the surface field. The dependence is reversed: the smaller the intensity of the field, the greater the screening effect by the adsorbed molecules.[27]

Similar considerations apply in the case of some other nonsulfide minerals.[28,29] Pyrite and arsenopyrite have also been the object of considerable study at the Institute of Mining in Moscow, and three methods[30,31] have been proposed for their separation. In the last analysis these methods rest on the crystal-chemical differences in the structure of the minerals.

Influence of Oxygen and Oxydizers on Selective Flotation of Pyrite and Arsenopyrite as a Function of Their Crystal Structure: In certain cases oxygen and the oxydizing chemicals can facilitate the selective separation of minerals with different crystal structure but similar flotation properties.

In modern practice selective flotation of such sulfide minerals as pyrite and arsenopyrite has not found complete practical solution. Both for industry and flotation theory, therefore, the selection of pyrite and arsenopyrite is of great importance, because with very similar flotation properties these minerals differ substantially in structure.

In the crystal structure of pyrite the ions of iron seem to be separated by ions of sulfur. The sulfur ions, being situated in pairs close to every ion of iron, are mainly concentrated at the surface and at the intersections of the crystal lattice of the minerals.

As to the morphologic peculiarities of crystals, arsenopyrite is classed with the markezite row. However, x-ray study shows certain deviations from this structure. The ideal structure of this mineral is monolithic; it becomes rhombic only when it is doubled. In the crystal structure of arsenopyrite-markesite the situation and the bonds of atoms are much more complex for arsenopyrite than for pyrite. In the former, every atom of iron has six neighbors in the corners of a somewhat displaced octahedron; one face of the octahedron is a triangle composed of three atoms of sulfur, whereas the opposite face is composed of three atoms of arsenic. An atom of sulfur is surrounded by three atoms of iron and one of arsenic, which are situated in the corners of a somewhat elongated tetrahedron. Accordingly, an atom of arsenic is surrounded by three atoms of iron and one of sulfur. The peculiarities of the crystal structure of pyrite and arsenopyrite reveal the different influence of oxidation upon them. The position of atoms of sulfur in pyrite on the faces and on the ribs of cells makes them more accessible for the joining of oxygen. In the first phase of oxidation it probably occurs without the destruction of the crystal lattice of the mineral.

Alteration of the ionic content of the aqueous part of the pyrite pulp which follows is associated with the dissolving of the oxidation products which takes place owing to weakening of the bond between the same adjacent atoms of sulfur and iron. However, the break-off of separate atoms apparently does not completely destroy the general structure of the mineral's crystal cell, and its newly revealed surface remains capable of interaction with flotation reagents. This is why even a protracted influence of oxidation does not depress floatability of the pyrite. As for the arsenopyrite, which has a more complicated structure both as regards its content and composition, the sulfur atoms are within the crystal cell in complex interrelation with the atoms of iron and arsenic and, consequently, are not accessible to oxygen influence in the first stage of oxidation. Hence, oxidation of arsenopyrite proceeds more slowly than that of pyrite. The processes of deep oxidation caused by prolonged oxidation in its more active form change the mineral structure by breaking the sulfurous and arsenic bonds. The arsenic ion is oxidized predominantly with the formation of arsenic oxide groups which remain on arsenopyrite surface. The study of flotation properties of pyrite and arsenopyrite has helped to establish the difference in their behavior under definite conditions of selective flotation, outlining three methods of separating pyrite and arsenopyrite: 1) use of oxidizers as reagents which differentiate the floatability of these minerals, 2) selective flotation in lime media with preliminary activation by $CuSO_4$, and 3) selective flotation with ammonium compounds and lime.

The third of these methods depends on the use of a relatively high concentration of ammonium chloride to favor flotation of pyrite and discourage flotation of arsenopyrite. This experimental result is corroborated by measurements of xanthate adsorption on both minerals made through xanthate tagged with S-35.

SUMMARY

In this necessarily very brief review of a very large research program, the writer has touched only the highlights. These include the following ideas:

1) Gases play an important chemical role in flotation. The behavior of minerals in particular depends on the concentration of oxygen in solution.

2) The effect of oxygen is not limited to sulfide minerals but, surprisingly, enters also into the behavior of nonsulfides.

3) The crystal structure of minerals, sulfides and nonsulfides alike, is a major factor in their response to variations in oxygen level.

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EXPERIMENTS IN CONCENTRATING IRON ORE FROM THE PEA RIDGE DEPOSIT, MISSOURI

by M. M. FINE and D. W. FROMMER

Mineral dressing research showed that iron concentrates of commercial quality could be produced from the Pea Ridge deposit near Sullivan, Mo. Magnetic separation and flotation, on a laboratory scale, yielded concentrates ranging from 60.0 to 71.4 pct Fe at recoveries of 89.9 to 97.6 pct.

Early in 1957 St. Joseph Lead Co. announced discovery of three new centers of iron ore deposition in east central Missouri.[1] The discovery resulted from exploratory drilling in the vicinity of a magnetic anomaly or *high*. A deep hole, drilled into the Pre-Cambrian porphyry to determine the cause of anomaly, penetrated a magnetite-rich deposit.[2]

The magnetic surveys that revealed the anomalies were begun by the Missouri Geological Survey in 1929 and were continued into the 1930's.[3] At that time an area in Crawford County known as the Bourbon magnetic high was located. A second anomaly, similar in size and intensity, was discovered about six miles to the northeast in Franklin County in the vicinity of Sullivan. This area was not studied in detail until the early 1940's.

To determine the cause of the anomalies, the USBM drilled four holes at the Bourbon site in 1943-1944.[4] The most productive hole penetrated four mineralized zones having a total thickness of 127.5 ft. These zones contained magnetite in rhyolite porphyry at depths between 1600 and 2000 ft. Additional iron ore below 2000 ft was considered a possibility at the time.

Some years later, the USGS, Missouri Geological Survey, St. Joseph Lead Co., and others cooperated in sponsoring an aeromagnetic survey of the Bourbon-Sullivan area which resulted in discovery of a third high, about eight miles distant at Pea Ridge. This deposit, with an estimated reserve of 50 to 100 million tons of iron ore, is considered the main find, although other orebodies in the same general area are of definite interest.[4]

When this discovery was announced, St. Joseph Lead also revealed plans to exploit the deposit jointly with Bethlehem Steel Co. A new corporation, Meramec Mining Co., was established, and 1962 was set as the target date for completing physical plant.[2] Annual production goals of 2 million tons of beneficiated and pelletized iron ore were proposed. Preparation of the site for shaft sinking was begun in June 1957.

Orebody and Character of Samples: Drilling to date has established the Pea Ridge orebody as a crescent-shaped pipe about 300 ft wide. Some drillholes were bottomed at 2800 ft in very good iron ore.[2] The orebody is predominantly magnetite, although hematite and a mixture of hematite and magnetite occur in the upper areas and peripheral walls.

Table I. Partial Chemical Analyses of Composite Samples

Number	Character	Analysis, Pct Fe	P	SiO_2	S	Cu
1	Medium grade mixed hematite and magnetite	40.3	0.87	27.2	0.35	0.04
2	High grade mixed hematite and magnetite	59.6	1.14	5.3	0.52	0.02
3	High grade magnetite	64.1	0.75	4.1	0.37	0.03

Drillhole cores representing ore from different areas and depths of mineralization were grouped into composites for testing. These composites consisted of: 1) medium grade mixed hematite and magnetite, 2) high grade mixed hematite and magnetite, and 3) high grade magnetite. The medium grade ore came from three holes with depths of 1350 to 1900 ft. The high grade mixed hematite and magnetite cores came from the 1790 to 2812-ft interval of one hole.

Partial chemical analyses of the three samples, given in Table I, reveal that all are excessively high in phosphorus and sulfur, and sample 1 is

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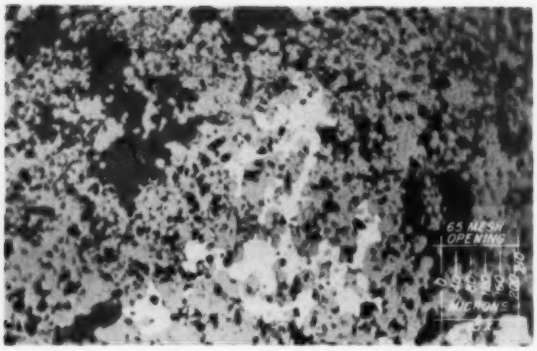

Fig. 1—Pyrite and gangue in magnetite. 75X, ¼ reduction.

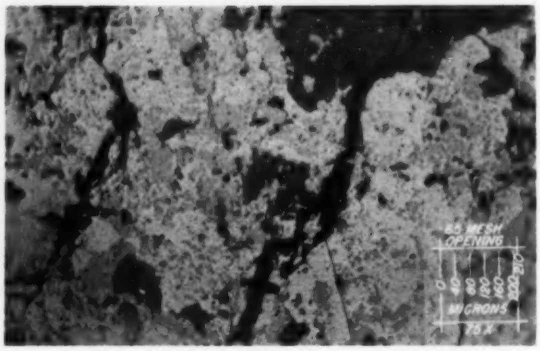

Fig. 2—Magnetite, hematite, gangue. 75X, ¼ reduction.

over the silica specification as well. It is notable, too, that samples 2 and 3 would be direct-shipping ore were it not for the phosphorous and sulfur contamination.

A petrographic examination of sample 1 revealed hematite and magnetite in a gangue comprised primarily of orthoclase, quartz, and chlorite. Smaller amounts of apatite, pyrite, chalcopyrite, and traces of tremolite and calcite were noted. Hematite and magnetite were in sample 2, of course, and the gangue minerals included quartz, feldspar, apatite, pyrite, and chalcopyrite. Finally, magnetite was the essential constituent of sample 3, accompanied by small quantities of the same gangue minerals noted for sample 1.

Figs. 1 through 3 are micrographs of polished sections made from random fragments at a magnification of 75X. In Fig. 1 the light-colored material is pyrite, the gray is magnetite, the dark gray is gangue (in this case, primarily apatite), and the black areas are depressions in the section. The grain size may be gaged by comparison with the photographic insert of a 65-mesh opening. Fig. 2 shows hematite (light gray) as an alteration product of magnetite (medium gray), and the gangue is represented by the dark gray areas. The light-colored area in the center of Fig. 3 is a grain of chalcopyrite; the light gray and the medium gray areas are hematite and magnetite, respectively; and the dark gray area is apatite. The micrographs, particularly Figs. 1 and 2, are eloquent evidence of the intimate association of iron oxides and gangue in the samples.

BENEFICIATION RESEARCH

Magnetic Separation: Since most of the Pea Ridge samples were comprised of magnetite, the initial experiment was recovery of iron by magnetic methods. From the intergrowth of minerals revealed in polished sections, it was anticipated that fine grinding would be required for adequate liberation. Accordingly a portion of sample 3 was ground to −48 mesh and sized by screening, and the individual fractions were separated in a Davis magnetic tube. These results, presented in Table II, showed a gradual reduction in phosphorous analysis of the magnetic concentrates down to 325 mesh and then a sharp drop to a value about half that of the coarser fractions. This suggested that the phosphorous content of magnetic concentrates produced from sample 3 would be a function of the quantity of −325 mesh material in the feed. In Table II it is evident that sulfur was also efficiently eliminated by fine grinding and magnetic separation.

Subsequent wet separations on a laboratory drum-type machine showed that grinding to at least −100 mesh was required to yield magnetic concentrates with a satisfactory iron-to-phosphorus ratio. A nominal 100 mesh grind created about 50 pct −325 material. The ground samples were given a two-stage magnetic treatment at a current of 0.5 amp; tailings from the first stage were thickened and passed through the unit a second time to recover additional magnetic values.

Table II. Sample 3, Separation in Davis Tube

Product	Mesh	Weight, Pct	Analysis, Pct				Pct of Total	
			Fe	P	SiO_2	S	Fe	P
Magnetic	−48 +100	22.6	69.0	0.35	2.7	0.17	24.2	10.1
Nonmagnetic	−48 +100	1.6	10.0	6.44	—	—	0.2	13.2
Magnetic	−100 +150	12.2	68.7	0.35	2.0	0.11	13.0	5.4
Nonmagnetic	−100 +150	1.1	8.8	7.08	—	—	0.2	10.0
Magnetic	−150 +200	12.0	69.7	0.31	1.8	0.07	13.0	4.7
Nonmagnetic	−150 +200	1.0	8.6	7.08	—	—	0.2	9.1
Magnetic	−200 +325	11.7	70.7	0.27	1.3	0.06	12.9	4.1
Nonmagnetic	−200 +325	1.2	12.5	6.77	—	—	0.2	10.4
Magnetic	−325	32.1	71.2	0.15	0.80	0.03	35.5	6.1
Nonmagnetic	−325	4.5	7.9	4.64	—	—	0.6	26.9
Composite		100.0	64.3	0.78	—	—	100.0	100.0

Table III. Sample 3, Magnetic Separation in Drum-Type Separator

Product	Weight, Pct	Analysis, Pct				Pct of Total	
		Fe	P	SiO_2	S	Fe	P
1st magnetic	85.9	71.5	0.21	1.1	0.047	94.0	21.3
2nd magnetic	3.0	68.0	0.45	2.8	0.14	3.1	1.5
Nonmagnetic tailings	11.1	17.0	5.87	—	—	2.9	77.2
Composite	100.0	63.3	0.84	—	—	100.0	100.0
Total magnetic concentrate	88.9	71.4	0.22	1.2	0.05	97.1	22.8

The results of separating sample 3 magnetically at −100 mesh are given in Table III. Recovery of 97.1 pct Fe was effected in a product grading 71.4 pct Fe, 1.2 pct SiO_2, 0.22 pct P, and 0.05 pct S. The second magnetic product (Table III) was made part of the total concentrate without appreciable detriment to the overall analysis, although it contained many locked magnetite-gangue grains. Their pres-

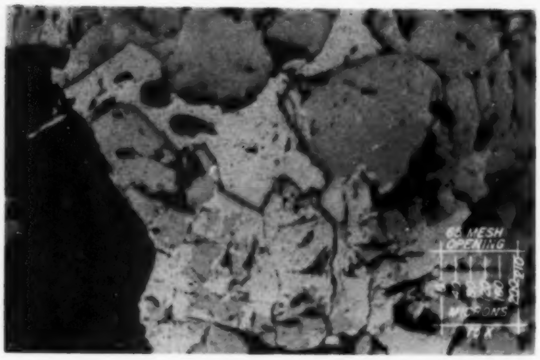

Fig. 3—Magnetite, hematite, chalcopyrite, and apatite. Micrograph reduced about one-fourth for reproduction.

ence, however, emphasized the importance of sufficient grinding to prevent too many of these highly magnetic middling particles from reporting with the concentrate.

Flotation: Samples 1 and 2, with hematite contents of about 55 and 40 pct respectively were, in the natural state, only partially responsive to low-intensity magnetic separation. Accordingly, research was undertaken to develop flotation procedures for concentrating these materials. With regard to major impurities, there was no difficulty in removing sulfur by conventional sulfide flotation or some variation of this method. Silica was a problem only on sample 1, and there the difference in floatability of iron oxides and the quartz-silicate gangue was so great that the gangue was readily rejected. Phosphorus was much more of a problem, since both the crystalline apatite and iron oxides were receptive to anionic collectors.

Research on flotation of iron ores has been under way in industrial and government laboratories for many years, and technical literature contains a profusion of related experimental data. Most of the research has been devoted to separation of siliceous ores, for which at least three solutions have been proposed and one (fatty acid flotation) applied commercially.[7] One reference[8] to flotation of iron ore containing apatite suggests that in an acid pulp fluorides or fluosilicates should help depress the phosphate mineral during anionic flotation of iron oxides. A similar combination of reagents has been notably successful, on an industrial scale, in floating ilmenite in the presence of apatite.[9]

Much the same approach was made with the samples of Pea Ridge ore, but it was soon evident that although the reagent combination of sodium fluoride and sulfuric acid was an excellent depressant for quartz and silicate minerals, it did little to discourage flotation of apatite. The iron oxides, on the other hand, could be depressed by alkaline silicates while floating apatite with moderate quantities of fatty acid or tall oil soaps. Iron minerals were easily reactivated with larger quantities of anionic collector. On this basis samples 1 and 2 were successfully concentrated.

The medium grade ore, sample 1, was wet-ground to −65 mesh and conditioned with soda ash and sodium silicate. Apatite and the sulfides were then promoted in succession with a tall oil soap and xanthate, respectively. The pulp was settled and thickened by decantation. A small amount of slime,

Table IV. Sample 1, Flotation

Metallurgical Results

Product	Weight, Pct	Analysis, Pct				Pct of Total	
		Fe	P	SiO_2	S	Fe	P
Apatite concentrate	6.6	8.6	11.5	—	—	1.4	83.0
Apatite middling	3.0	33.0	0.47	—	—	2.5	1.5
Sulfide concentrate	0.9	37.3	0.44	—	—	0.9	0.5
Sulfide middling	1.2	26.3	0.31	—	—	0.8	0.5
Iron concentrate	58.9	60.0	0.19	9.8	0.075	89.9	12.1
Iron middling	6.6	10.0	0.06	—	—	1.7	0.3
Tailing	20.2	4.0	0.04	—	—	2.0	0.9
Slime	2.6	11.8	0.42	—	—	0.8	1.2
Composite	100.0	39.3	0.91	—	—	100.0	100.0

Operating Data. Apatite and Sulfide Flotation

Reagents	Pounds Per Ton of Crude Ore						
	Apatite Flotation				Sulfide Flotation		
	Conditioners 1	Conditioners 2	Rougher	Cleaner	Conditioner	Rougher	Cleaner
Sodium carbonate	2.0						
Sodium silicate	2.0						
AC 710*		0.5					
Secondary butyl xanthate					0.10		
Aerofroth 65*					0.02		
pH	10.2						
Time, min	5	5	5	5	5	5	3

Operating Data. Iron Flotation

Reagents	Pounds Per Ton of Crude Ore				
	Conditioners 1	Conditioners 2	Rougher	Cleaners 1	Cleaners 2
Sulfuric acid	0.67			0.20	0.20
Sodium fluoride	0.25			0.08	0.08
Pamak No. 1**		0.90			
pH	7.5				
Time, min	5	10	5	5	5

* American Cyanamid Co.
** Hercules Powder Co.

Table V. Sample 2, Flotation

Metallurgical Results

Product	Weight, Pct	Analysis, Pct				Pct of Total	
		Fe	P	SiO_2	S	Fe	P
Apatite concentrate	9.5	11.8	11.25	—	—	1.9	85.4
Apatite middling	1.9	50.8	0.79	—	—	1.6	1.2
Sulfide concentrate	0.9	39.0	0.58	—	—	0.6	0.4
Sulfide middling	1.1	49.0	0.54	—	—	0.9	0.5
Iron product	86.6	65.5	0.18	6.4	0.067	95.0	12.5
Composite	100.0	59.7	1.25	—	—	100.0	100.0

Operating Data

Reagents	Pounds Per Ton of Crude Ore						
	Apatite Flotation				Sulfide Flotation		
	Conditioners 1	Conditioners 2	Rougher	Cleaner	Conditioner	Cleaner	Rougher
Sodium carbonate	2.0						
Sodium silicate	2.0						
Sodium oleate		0.5					
Secondary butyl xanthate					0.12		
Aerofroth 65					0.02		
pH	10.2						
Time, min	5	5	5	5	5	5	5

containing less than 1 pct of the total iron, was decanted in the process. The thickened slurry, at about 40 pct solids, was conditioned with sulfuric acid and sodium fluoride to establish a near-neutral pH. A rougher concentrate was floated with a refined tall oil fatty acid and cleaned twice to produce a finished concentrate.

Details and results are presented in Table IV. Recovery of 89.9 pct was attained from sample 1 at a grade of 60.0 pct Fe, 0.19 pct P, 9.8 pct SiO_2, and 0.075 pct S.

The high grade mixed ore, sample 2, was treated similarly, but since this material contained little silica it was necessary only to float the apatite and sulfides to yield a nonfloat product of acceptable quality. Table V gives the results and operating data in flotation of sample 2, from which 95.0 pct recovery was effected at a grade of 65.5 pct Fe, 0.18 pct P, 6.4 pct SiO_2, and 0.067 pct S.

As a sidelight of the flotation research it was discovered that in upgrading the iron, the apatite could be recovered as a byproduct of possible commercial value. In some tests, recoveries of 75 to 80 pct of the phosphorus were effected at grades of 66 to 71 pct tricalcium phosphate (BPL). The Pea Ridge iron ore deposit, it has been announced, will be brought into production within the next five years at an annual output of 2 million tons of concentrate. At that rate, Pea Ridge could also be a source of some 100,000 tons of apatite concentrate per year. Apatite does not compete with phosphate rock for production of wet-process superphosphate because it is less reactive with sulfuric acid, but it should be a satisfactory raw material for electric-furnace phosphorus.

If a market is developed for the apatite, the logical flowsheet for all three types of Pea Ridge ore would consist of magnetic separation followed by flotation, which would insure maximum recovery of apatite and both iron oxides. Research to confirm this premise was conducted on samples 1 and 2 under this agreement and products of satisfactory quality were recovered. Recoveries were somewhat lower than those presented here, but only because time limitations prevented achieving optimum results.

The work on which this report is based was done under a cooperative agreement between the U. S. Bureau of Mines and St. Joseph Lead Co.

The authors wish to acknowledge the complete cooperation of St. Joseph Lead Co., which supplied not only the samples with which the research was conducted, but also funds for the experimental and analytical work.

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Discussion of this article sent (2 copies) to AIME before April 30, 1959, will be published in MINING ENGINEERING.

PERMISSIBLE-TYPE DUST COUNTER FOR COAL MINES

by ALBERT L. THOMAS, JR., and SABERT OGLESBY, JR.

Until recently, probably the best means of sampling airborne dusts has been the impinger method. Dust-laden air is drawn into a sampling tube, and the particulate matters separated from the air and collected in the liquid in the impinger. A sample of the suspension is prepared on a slide and counted through a projection microscope.

This method is simple and reliable, but it does suffer drawbacks. Since the collected particles must be counted visually, results of the dust count are not immediately available at the time the sample is taken. This seriously impedes studying the effectiveness of dust inhibitors, such as water sprays, or making adjustments in equipment. Greatest disadvantage of all, however, is the tedious task of counting particles in the sample. With the growing desirability of taking more samples, the effort involved in making these counts is a real factor in the cost of dust control. Finally, it is generally recognized that impinger dust counts can be related to health hazard only for specific kinds of dust and under similar conditions. This is because the rather complex mechanism of dust damage depends so much on size distribution of the dust. Counts determined by impinger samples cover a limited size range that does not necessarily corre-

A. L. THOMAS, JR., is Head of the Instrument Development Section and S. OGLESBY, JR., is Head of the Engineering Division, Southern Research Institute, Birmingham, Ala. TP 4797F. Manuscript, March 28, 1958. New York Meeting, February 1958. AIME Trans., Vol. 214, 1959.

late with retention by the lungs and upper respiratory tract.

Several other methods for making dust counts in the mine offer advantages over impinger sampling, especially in giving instantaneous readings of the dust concentration or dust level. These methods will not be evaluated here, but it is generally recognized that each has its own particular shortcoming.

In considering development of a counter for use in coal mines, the writers set forth the following desirable properties:

1) It should read number of particles rather than mass concentration. This is important, since it is recognized that it is primarily the number of irritation centers that create respiratory damage, rather than the total quantity inhaled.

2) It should read dust concentration directly. This is a requisite for any improved dust sampling method because of the labor associated with visual counts.

3) It should be capable of reading particles down to the smallest retained in the lungs, although it will not be used at that sensitivity for comparison with counts made by the impinger method.

4) It should be possible to correlate data obtained with results of impinger sampling methods. This is important because of the great amount of work already done in attempting to get agreement between potential hazard and dust levels as measured by impingers.

5) It must be a type permissible in gassy mines.

The type of counter the present investigators are working on is a particle photometer operating on the forward scattering of light from an aerosol or dust particle. The optical layout of the counter, shown in Fig. 2, is composed of a light source, a system of lenses and stops, a photoelectric tube, and the associated electronic circuitry. The light source in this counter is a No. 13 flashlight bulb.

The lens system gives a condensed image of the light filament just over the end of the tube shown in the lower part of Fig. 2; the beam is then stopped in the lower light trap. The photoelectric tube *looks* across the nozzle and the light path into a dark, velvet-lined chamber. The area above the nozzle that is illuminated by the lamp and the area seen by the phototube form the sensitive volume. Now if this volume contains no particles, the phototube will see nothing but the dark chamber. However, if there is a particle in the view volume, light will be scattered from the particle onto the sensitive cathode of the phototube. If the particle is moving there will be a pulse from the phototube, the amplitude of the pulse being generally proportional to the size of the particle, for constant illumination and phototube characteristics.

Fig. 1, showing the view volume for a particle counter, was made with a dense smoke passing through the nozzle and with considerable background light. The camera in this instance replaces the phototube and the smoke is considerably denser than in normal counting, but if sensitive enough film were available, it would be possible to record the trail of a particle across the illuminated area.

The phototube is sensitive enough to record the light from the small particles, so that in practice the electrical pulse from the phototube, rather than the film, is used to record or count the particles.

This particular optical arrangement may be used not only for counting particles but, with suitable circuitry, can be made to give the particle size as well. If the particles are homogeneous, and if the light intensity and phototube characteristics do not change, the size or height of the electrical signal from the phototube will be proportional to particle size. It is relatively easy then to show not only the particle count, but also to draw a histogram of particle size distribution. This is not a feature of the present model counter, but it does have a number of applications.

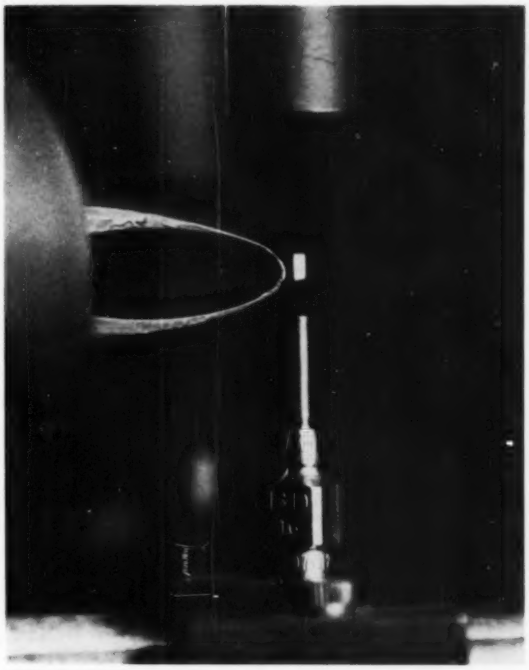

Fig. 1–Smoke through nozzle.

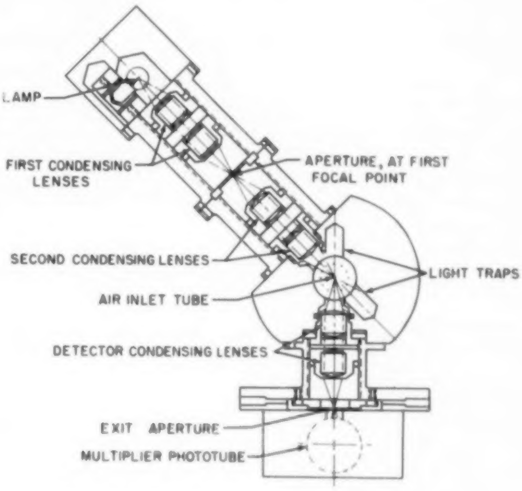

Fig. 2–Optical layout of particle counter.

The relationship between the height of the electrical pulse and the particle size is important in

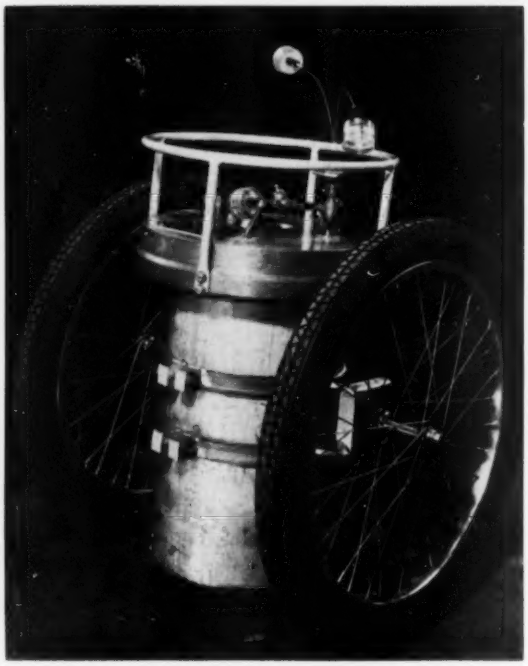

Fig. 3–View of instrument in case.

other regards, because it establishes the minimum particle size that can be counted. This minimum size depends on a number of factors in design and layout of the counter's optical system, on the optical properties of the particles being counted, and on characteristics of the phototube and electronic circuitry. To be counted, the pulse amplitude must be great enough so that the electronic circuitry can distinguish it from the random current in the phototube. This will be discussed later in more detail.

The electrical circuit of the counter is shown in Fig. 4 in block diagram form. Chief purpose of the electronic circuitry is to convert the pulses from the phototube into a voltage proportional to the number of pulses per second. The voltage is displayed on a meter that is calibrated in particles per second.

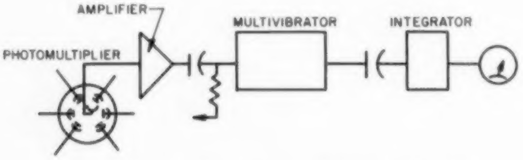

Fig. 4–Block diagram of electrical circuit.

The first block in the diagram is an amplifier, which increases the height of the pulses so that they can be handled by the remaining circuits. The output of the amplifier is fed to a multivibrator circuit. Each pulse from the amplifier above a given minimum voltage triggers the multivibrator, giving an output pulse of constant height regardless of the height of the input pulse.

Finally, the output of the multivibrator is fed to an integrating circuit whch gives an output voltage proportional to the number of pulses per unit of time. This voltage is indicated by the panel meter inside the top of the counter case.

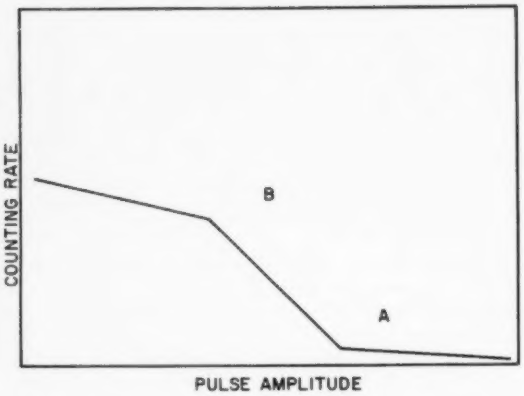

Fig. 5–Counting rate vs discriminator setting.

It is apparent from the operation of the counter that not more than one particle should be in the view volume at one time. Otherwise the output would be a single pulse of larger size than is indicated by a single particle, resulting in erroneously low dust counts. Consequently, in order to count the heavy concentrations found in the mines, input must be greatly diluted.

Dilution of aerosols or dusts is, of course, a very tricky problem. If it is not performed accurately the apparent counts will be in error. Also, the dilution system should not alter the resistance to flow; otherwise the flow rate would change with dilution. Finally, any dilution method should not alter the particle size distribution. It all sums up that the diluted sample should be representative.

In the Southern Research Institute counter dilution is accomplished by a special value that mixes filtered air with the unfiltered air, then samples the mixture and discards the excess isokinetically. This is done in a novel device for which a patent has been applied. Accurate dilutions up to 10/1 are possible with this valve.

The basic counter is capable of counting a particle density of 10 million per cu ft of air with no dilution. Consequently, a maximum density of 100 million per cu ft can be counted with a 10/1 dilution; greater dilutions approaching infinity are possible, though less accurate.

As mentioned previously, one of the requirements was that the counter be approved for use in gassy mines. The case for housing the counter, therefore, had to meet USBM requirements of containing an explosion set off within the case. To meet these requirements, the case was made of a heavy aluminum pipe. A brass, screw-type lock ring fastens the instrument to the case.

The most difficult problems in development of the dust counter were in adapting it as a field instrument. Basic techniques for the counter as a laboratory device had previously been worked out. As the instrument was developed, it was decided to have it completely self-contained with all batteries inside the case. It is also important that these batteries have a long life to reduce maintenance problems. This means low current drain. On the other hand, it is important that light intensity be high in the view volume. An ideal light source would be a ribbon filament lamp with filament

dimensions about ⅛ in. square. This filament could be focused by the lens system on the view area and would give the best illumination pattern. This would give the best signal-to-noise ratio and would permit counting the smallest particle size. Unfortunately such a lamp is not available, and if it were, the voltage-current characteristics would probably not be suitable. The No. 13 flashlight lamp used as a compromise gives low current drain and enough illumination to count particles down to about 0.3μ, although the illumination is not as uniform as from a ribbon lamp.

Calibration of the dust counter is done by sampling an aerosol of known particle size. Polystyrene latex is available from Dow Chemical Co. in various measured sizes from 0.1 to 2.0 μ, and the range of particle size in each sample is extremely narrow. The procedure for setting the sensitivity of the counter is to generate an aerosol of the desired minimum size to be counted and to vary the discriminator setting. Counting rate will change with discriminator setting, as shown in Fig. 5. The plateau on the curve indicates the setting at which all of the particles in this size range are being counted.

Two points on the curve, *A* and *B*, are used to indicate sensitivity of the counter. Point *A* is the discriminator setting at which the counter just begins to count the particles and point *B* the setting at which all the particles are being counted. Once these points are established for a given aerosol, the counter can be set to, and returned to, any desired sensitivity.

One difficulty in using polystyrene latex for calibration is that the reflectance for polystyrene is considerably different from that of coal dust. The minimum size coal dust that can be counted therefore is smaller than that of the polystyrene particle, for which the minimum size has not yet been determined. However, it has been established that the counter can be set to be sensitive to particles smaller than are countable with the usual projection microscope technique.

Fig. 6 compares readings of the particle counter with impinger counts to show agreement between

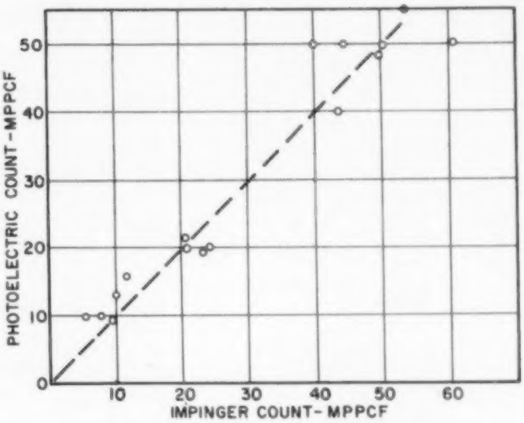

Fig. 6—Correlation of impinger and photoelectric data.

the two methods. Data for this curve were taken by simultaneously sampling coal dust from a chamber with the particle counter and with an impinger. Agreement is well within the variability of counts by impinger methods.

Before it can be declared operational, the counter must be tested extensively in the mine. To be successful, it must give long trouble-free service and survive rough handling. There must be a rapid and convenient way to check calibration in order to develop confidence in the counter's reliability.

Work has now reached the field trial stage. The counter is being used by the industrial engineering department at Alabama Power Co.'s Gorgas mine, where the mechanics are being checked as well as the optical and electrical features. Several factors have already been found to influence operation of the instrument, which is being modified accordingly. There is every hope that it will prove successful in coal mine dust sampling and that it will significantly reduce the cost of mine dust control.

Discussion of this article sent (2 copies) to AIME before April 30, 1959, will be published in MINING ENGINEERING.

Discussion

INTERGRANULAR COMMINUTION BY HEATING

by J. H. BROWN, A. M. GAUDIN, and C. M. LOEB, JR.

(MINING ENGINEERING, page 490, April 1958, AIME Trans., Vol. 211)

R. E. Carter *(General Research Laboratory, Schenectady, N. Y.)*—Brown, Gaudin, and Loeb in their study of intergranular comminution by heating attempt to find one explanation for all types of rock in terms of the properties of the phases present. The *only* necessary condition for intergranular cracking by heating is the setting up of high stresses at the grain boundaries.

Such stresses may be set up in multiphase isotropic mixtures if the thermal expansions of the phases are different, in non-cubic single phases because of the anisotropy of thermal expansion in the various crystallographic directions, and in materials containing quartz because of the α-β transformation at 575°C. In addition, for intergranular comminution by heating, the cooling and not the heating is most important.

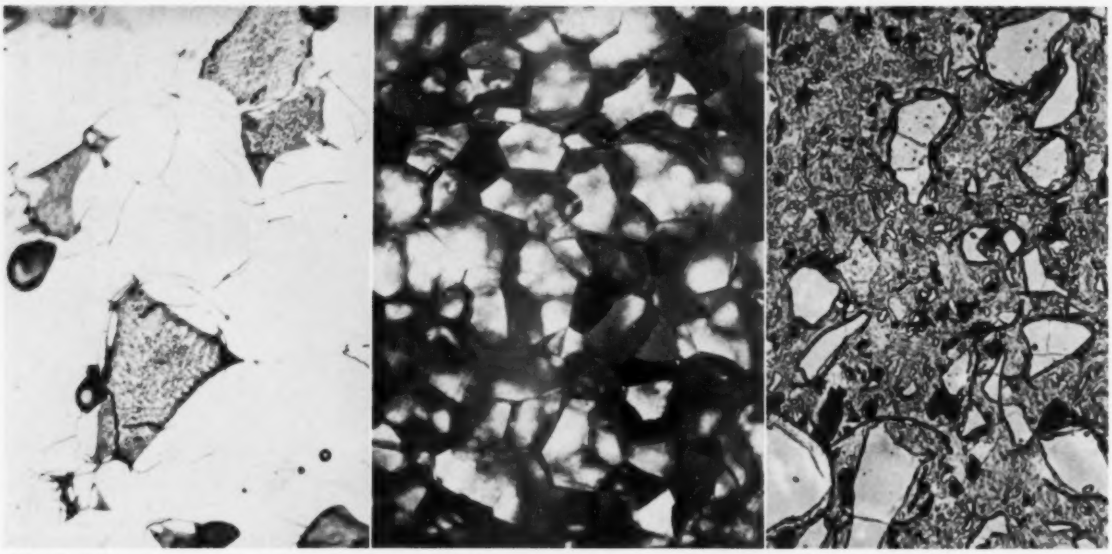

Fig. 9—Microcracks formed during cooling. (Photographs reduced approximately 15 pct for reproduction.) Section A (left)—Magnesium ferrite plus a Wüstite-like precipitate $SnCl_2$HCl etch. Reflected light. X500. Section B (center)—Single-phase alumina. Unetched. Transmitted light. X30. The intergranular cracks reflect light and make the grains visible. Section C (right)—Porcelain containing undissolved quartz. Phosphoric acid etch. Reflected light. X500.

Almost all the cracking must occur on cooling, since on heating part or all of the stress may be relieved by plastic flow, but on cooling plastic flow will be almost absent. An extension of this argument suggests that the faster the cooling the less the plastic flow and the greater the stress; hence the greater the possibility of intergranular cracking.

In Fig. 9 there are three sintered materials, all of which show intergranular or near intergranular cracking as a result of cooling from the sintering temperatures. Section A shows a cubic magnesium ferrite containing a cubic Wüstite-like precipitate. The precipitate has a higher coefficient of thermal expansion than does the ferrite phase and on cooling from 1400°C a tensile radial stress of 40,000 psi is set up at the interface if no relaxation occurs during cooling.[14] Section B shows a specimen of single phase alumina which R. L. Coble of the G. E. laboratory reports was sintered at 1800°C and then air-quenched. During cooling, stresses are set up at the grain boundaries because of the anisotropy of the thermal expansion of this rhombohedral material. The formation of the cracks may be observed with a binocular microscope using transmitted light. In many cases the cracking was accompanied by an audible ring.[15] Section C shows a porcelain containing large pieces of undissolved quartz. These cracks could not have formed during heating, since the material placed in the furnace was a mechanical mixture of quartz, feldspar, and clay which then reacted at the sintering temperature. The cracks must therefore have resulted from the high radial tensile stresses set up in the porcelain on cooling because of the volume contraction of the quartz particles during β to α transformation.

J. H. Brown, A. M. Gaudin, and C. M. Loeb, Jr. *(authors' reply)*—We wish to thank R. E. Carter for his contribution. His observation that the only condition necessary for intergranular cracking by heating is the setting up of high grain boundary stresses seems accurate to us, and we agree with his listing of several methods by which this could occur.

On the other hand, we cannot agree with his statement that for intergranular comminution by heating the cooling and not the heating is most important. It may be true that plastic flow will be more likely on heating than on cooling, but we are not aware of any indication that plastic flow rates for these materials are so highly temperature-dependent in this temperature range that it is particularly significant that a rock is either heating or cooling. Certainly the figures presented to support the hypothesis of cracking during cooling do not clarify the point, since in these specimens no structure was available to fracture during heating. We feel that Carter has specified still another method by which heat treatment prior to crushing may affect the rock. There is merit in the suggestion that fracture may occur when the cooling rate is high enough to cause large temperature differentials in the cooling specimen.

In support of our hypothesis that intergranular fracture occurs during heating to a significant extent, we refer again to the data presented in Figs. 2 and 5. If the cooling rate is important, it is difficult to explain why at least as great an effect, if not a greater one, was observed when the specimen was not allowed to cool at all before crushing.

A final test of the cracking-during-heating proposal was made in our laboratory recently. A specimen of granite about ½x½x3 in. was held by one end on a block to leave about 2½ in. of unsupported specimen. This cantilever was then loaded with a weight which we estimated as being 10 to 15 pct of the load required to fracture the piece in the unheated state. The entire system was then set into a furnace and a signal system by which the moment of fracture could be detected was attached to the specimen. The furnace was then closed and heated gradually. At a temperature of about 500°C *during heating* the specimen broke.

We do not deny the possibility that fracture may occur during cooling. We merely state that the fracture that occurs during heating was highly significant and we hope that a report on further work in this area which is now in preparation will contribute to this question.

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SME BULLETIN BOARD

Reports of Your Technical Society

UTAH REV(IEW)UE

The Utah Section has done it again—presented their third annual Miners' Review, that is. To find out about the hilarious goings on—and this picture—turn to page 340 for a pictorial report.

SME PREPRINT LIST

An up-to-date SME Preprint List is given on page 284. Corrections and additions have been made in the list published in the February issue. Complete ordering instructions are given.

MINING ENGINEERING salutes J. W. (Jerry) Woomer on page 287. The 1959 President of the Society of Mining Engineers took office in February.

News of Education—page 334

NEWSLETTERS . . .

- Coal page 339
- Rock in the Box page 337

ATTENTION . . . SME MEMBERS

Membership and the growth of SME continue to be an individual responsibility. Help strengthen your professional society by bringing a colleague to the next Local Section meeting. Introduce him to SME of AIME and help him fill out a membership application.

DIRECTORY CALL

SECOND NOTICE

Notice is hereby given to Society of Mining Engineers' members that Directory Listing cards are due to be returned to AIME by April 1. To ensure proper listing in the 1959 Members' Directory, send in your card promptly. No second notice will be mailed you. The SME Directory will be bound into the July issue of MINING ENGINEERING.

Coming Events: Annual Alaskan Conference
IndMD—Coal Fall Meeting

Change of Address
Membership Proposal Boxes

these forms are printed on page 344
for your convenience.

Education News

Western Michigan University

A new program in industrial engineering will be offered at Western Michigan University in the fall of 1959. The heavy concentration of industry in the area has prompted the action of the State Board of Education, on the suggestion of Paul V. Sangren, president of the university. Extensive work is already offered in basic sciences, plus time and motion study, industrial management, quality and quantity control, metallurgy, etc., so that the addition of these courses in applied mechanics will provide an industrial engineering course comparable to the curricula in the best engineering schools of the country. In 1960 additional courses and staff will be provided.

University of Wyoming

The Atomic Energy Commission has proposed a permit to the University of Wyoming in Laramie for the construction of an L-77 reactor for use in experimentation and training. The Commission will provide a grant of $135,408 to cover the cost of the reactor and equipment. The University will provide funds for its operation and a remodeled building for its housing on the campus. The solution-type reactor is designed to operate at a thermal power level of 10 kilowatts and will be built by Atomics International.

Colorado School of Mines

A contribution valued at $140,000 has been donated to the Colorado School of Mines Foundation Inc. by A. Hartwell Bradford, Los Angeles oil man. The gift, which consists of 2000 shares of Continental Oil stock, is the largest single contribution ever made to the eight year old Foundation. The Foundation directs a program aimed at constantly improving the School's faculty, scholarship, and fellowship programs.

• A Soil Mechanics Conference will be held on April 23 to discuss construction costs in the Rocky Mountain area. The program will feature five papers on theoretical and practical treatments of expansive soils. It is being co-sponsored by the Soil Mechanics and Foundation Div. of the Colorado Section of the American Society of Civil Engineers, and will be held in the Library of the School of Mines.

• About a third of the teaching faculty at the School of Mines is back in school learning how to program the new computor laboratory machines that can solve a large backlog of problems. An $11,000 analog computor was purchased two years ago with allocated state money, and last year the Colorado School of Mines Foundation paid $30,000 for a digital computor. As soon as there are enough trained people to provide problems for the machines to solve, thousands of manhours and dollars can be saved. John C. Hollister, head of the Dept. of Geophysical Engineering, estimated that it would have taken a year and a half to solve a problem on a hand calculator that the digital computor solved in an hour and a half. With trained men in each department, 85% of the preparation time in planning problems for the machines can be performed by the departments, allowing math teachers to concentrate on their undergraduate and graduate courses.

West Virginia University

The School of Mines at West Virginia University will offer the Ninth Annual Short Course in Coal Preparation at Morgantown, June 8 to July 17, 1959. Industrial leaders have credited the course with meeting a definite training need. Designed for men already employed in the cleaning and preparation of coal, it is also available to qualified students who wish to obtain four hours college credit. Non-credit students may register on arrival June 8, and credit students should apply in April to J. Everett Long, Committee on Admissions, West Virginia University.

Anchorage Will Host April Technical Forum

The Annual Spring Technical Forum in Anchorage, Alaska, will be co-sponsored by the Southwest Alaska Section and the Alaska Section from Fairbanks on April 3 to 5, 1959. Members of the Yukon Section of the Canadian Inst. of Mining, Metallurgical and Petroleum Engineers will also attend.

Technical papers will fill the first two days; field trips are scheduled for the third. Economic, technical, and political aspects of the Alaskan mining industry are the topics for the 3rd of April. Petroleum is the theme of the second session.

Harold Strandberg of the Strandberg Mines Inc. is the general chairman of the conference, in charge of coordinating the Forum committees. Committee members include: Pat Ryan; G. Chambers; B. Anderson; J. F. Homer; Locke Jacobs; M. Jasper; R. Richards; W. Robinson; B. Strandberg; D. Hill, and E. M. Case.

The Westward Hotel in Anchorage will be the headquarters for registration beginning Thursday, April 2. For further information write to: AIME, P.O. Box 2139, Anchorage, Alaska. All members of AIME and interested guests are invited to attend and urged to register immediately.

Officers of the Southwest Alaska Section include J. F. Homer, chairman; G. A. Gustafson, vice chairman; and Julius Moor, secretary-treasurer.

New Mexico, Arizona Hold Joint Field Trip

The Ninth Annual Field Conference of New Mexico Geological Soc. was held with the cooperation of the Arizona Geological Soc. in the Black Mesa Basin area of northeast Arizona, Oct. 16 to 18, 1958. A large caravan of 110 cars with 340 people composed the field trip, and all enjoyed the perfect weather.

Stratigraphic and structural features of the area were stressed in discussion of the geology along the way. The trip started from Gallup, N. M., and wound up at the Grand Canyon, with stops in between at Window Rock, Defiance Uplift, Petrified Forest, and Echo Cliffs.

At Hunters Point the explorers observed the Permian beds resting on pre-Cambrian rocks. They visited the non-swelling montmorillonite deposit near Sanders, Ariz., from which nearly 5,000,000 tons of clay have been shipped to oil refineries for use as a catalyst and a bleaching agent. In the Petrified Forest, the campers were fascinated by the agatized logs and the dioramas.

Lectures highlighted the excursion at various points. Vincent Kelley discussed the tectonics of the Black Mesa Basin and showed slides of the structures and formations. The climatology of the area also was discussed by Roger Anderson. And at the Oraibi High School, the caravan was welcomed by two officials of the Hopi Indian tribe.

The third day found the caravan crossing the San Francisco Mountains on its way to Sunset Crater for a study of glaciology and late volcanic action. The Wupatki Indian ruins were the next site of study, before arriving at the Grand Canyon where all of the Paleozoic section was described from Desert View lookout by E. D. McKee who then led part of the group into the Canyon on the Kaibab Trail.

The Guidebook of papers covering the area was edited by Roger Y. Anderson, University of New Mexico, and John W. Harshbarger, U. S. Geological Survey. Some of the papers included are: *Precambrian Rocks of Northern Arizona:* J. F. Lance; *Devonian System of the Black Mesa Basin:* D. S. Turner; *Pennsylvanian Paleogeography of Arizona:* Kay Havenor and W. D. Pye, and many more. This compact and broadly informative guidebook is available for $8.75 postpaid, from the New Mexico Bureau of Mines and Mineral Resources, Socorro.

The report of the field trip was prepared by Dorsey Hager and forwarded by Richard D. Holt, general chairman of the New Mexico Geological Soc.

World Geology Meeting To Convene in 1960

The Nordic nations will again play host to geologists from all over the world on Aug. 15 to 25, 1960. Plans are already well on their way for the future conference. Once before in the history of the meetings, which began back in 1878, Scandinavia was the setting for the geological study, when Stockholm was the host in 1910. The site is a classical region of geological science and a popular tourist center in Europe, so it is only fitting that this oldest of scientific congresses meet in Copenhagen.

The first meeting was held in Paris by American scientists who felt the need for international cooperation on questions of classification and geological terminology. There have been 20 meetings since that time in capitals all over the world.

National committees have been established in the five Scandinavian countries, and a preliminary organizing committee is headed by Arne Noe-Nygaard, professor at the University of Copenhagen. He will act as president and general secretary will be Theodor Sorgenfrei of the Geological Survey of Denmark.

New Officers Elected

At the 37th regular meeting of the Grand Junction Geological Society, held Monday night, Dec. 1, 1958, the names of the new officers for 1959 were announced. The new president is Robert H. Sayre, Jr., consultant. Vice president is Raymond C. Robeck, also a consultant, and Philip Dinnerstag, Climax Uranium Co., is secretary. The treasurer is Don R. Hill of the Atomic Energy Commission. Two new councilmen are Richard J. Frost of Shell Oil Co. and Robert H. Toole of the A.E.C.

Four new members of the program committee, which is headed by the vice president, were also appointed: John G. Barry and M. A. Leaks of the Atomic Energy Commission; Max R. Hembree of California Co.; and Heyward M. Wharton of Union Carbide Nuclear Co.

MBd'ers digest

The selection of candidates for the Richards Award has occasioned a restudy of nominating procedures which MBD membership should give some consideration. A major problem is that many men who would be worthy candidates are not being placed in nomination. A greater number of deserving candidates could be placed on the Active List and become eligible for the Award.

The Richards Award Committee is appealing to you, the membership in MBD, to review the accomplishments of your associates and, if you know of a worthy candidate, to take the time and trouble to place his name in nomination. It is an honor to receive the Richards Award and it is the duty of MBD members to see that no worthy candidate is overlooked.

Any member of the Institute may initiate a nomination for the Award by submitting the candidate's name in writing to the Chairman of the Richards Award Committee together *with a complete professional and industrial record of the candidate, and a comprehensive statement of the particular achievement for which the candidate is nominated to receive the Award.*

The majority vote of the Richards Award Committee determines if a nominee's name is to be placed on the Active List where it must remain for one year before being eligible to receive the Award. After three years, names are dropped from the Active List.

The Award itself is to recognize achievement in any form which unmistakably furthers the art of minerals beneficiation in any of its branches. There are no limitations regarding nationality, membership in the Institute, or otherwise, except that younger men up to 45 are to be favored.

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ROCK IN THE BOX

Mining & Exploration Division

MINNESOTA SECTION ANNUAL MEETING CONVENE IN JANUARY

The Minnesota Section held one of the larger AIME meetings, and one of the most important in the iron ore field, on January 12 in Duluth. Their 32nd annual meeting boasted a record attendance of 649. In the last few years registration has grown from 430 in 1956, 573 in 1957, to 623 in 1958, showing an increasing interest in the future scope of iron ore. Good weather and driving conditions helped to encourage this fine turnout, and an excellent program rewarded the miners who attended.

The mining industry in 21 states, plus provinces in Canada, was well represented at the meeting, with members from as far away as Florida and Alabama. They came also to attend the two-day Mining Symposium of the University of Minnesota Center for Continuation Study.

A business meeting opened the conference, with Stephen E. Erickson, Section chairman, welcoming the registrants. Mr. Erickson is director of beneficiation for The M. A. Hanna Co. in Cooley, Minn. He complimented the committeemen who were responsible for planning the meeting, then called on secretary-treasurer Robert L. Bennett for his report. Mr. Bennett is assistant manager of research for Oliver Iron Mining Div. of U. S. Steel in Duluth.

William R. Van Slyke, chairman of the nominating committee, and range metallurgist for The Cleveland-Cliffs Iron Co., Taconite, Minn., gave his report of recommendations that were then voted upon by the members present. Election of 1959 officers composed the main portion of the meeting.

New Elections

The new chairman of the Minnesota Section is Robert J. Linney, executive vice president of Reserve Mining Co. Mr. Linney attended Yale University and began as a mining engineer in 1929 for Republic Steel's Port Henry, N. Y., operations. In 1950 he joined Reserve at Silver Bay, Minn., as manager of operations, and was elected executive vice president in 1958.

Kenneth E. Marklin was elected first vice chairman at the meeting. He is a metallurgist for Pickands Mather & Co. in Hibbing and served as program chairman for the Section's annual meeting.

Second vice chairman is Leon D. Keller, district manager for Dorr-Oliver Inc. with headquarters in Virginia, Minn. He is also chairman of the Minerals Beneficiation Subsection.

Third vice chairman, John J. Foucault, is manager of Minnesota Mines for The Cleveland-Cliffs Iron Co. in Hibbing.

Robert L. Bennett will keep his position as secretary-treasurer in accordance with the usual procedure of the Section of having a two-year period for the job.

After the elections there was discussion concerning the national AIME meeting, with suggestions for inviting the Institute to meet in the Lake Superior region in 1960. The MBD Subsection arranged a successful meeting in 1953 and it is hoped that another excellent program could be planned for the future.

Technical Sessions

The theme for the conference concerned the iron and the steel industries in Russia and the morning technical session was based on a trip to the USSR that 19 men had made in 1958. Ralph W. Marsden, director of mineral investigations for Oliver Iron Mining Div., acted as chairman of the session, introducing two speakers who had been to Russia.

Everett L. Joppa, general manager of Pickands Mather & Co., Duluth, was the first speaker on the topic *Iron Mining in Russia*. D. N. Vedensky, director of research and development for The M. A. Hanna Coal Co. in Cleveland, discussed *Russian Beneficiation Methods and Plants*. This session was held in the Norshor Theater.

Chairman Erickson presided at the luncheon in the ballroom of the Hotel Duluth. He introduced the guest speaker, William K. Montague, prominent Duluth attorney, who described the history and current trends in *Ad-Valorem Taxation of Mines in Minnesota*.

In the afternoon T. L. Joseph, professor at the University, presided over the session which reconvened next door at the theater. The meeting began with a few reels of film taken during the Russian trip by E. L. Joppa. He related events of the trip while commenting on the general interest scenes in the films.

The first technical paper of the afternoon was on *Steel Operations in Russia*, presented by J. H. Strassburger who is assistant vice president of Weirton Steel Co.

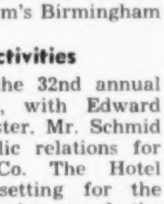
A final paper discussed *Direct Reduction of Iron Ores in the R-N Plant*. J. S. Breitenstein, vice president in charge of administration for the R-N Corp., New York, outlined operations at the firm's Birmingham plant.

Banquet Activities

In the evening the 32nd annual banquet was held, with Edward Schmid as toastmaster. Mr. Schmid is director of public relations for Reserve Mining Co. The Hotel Duluth was the setting for the festivities. Past chairmen of the Minnesota Section were awarded gold *Past President* pins, with those present receiving their honors at the banquet, and those who were unable to attend receiving theirs by mail. Stephen E. Erickson, chairman of the group for 1958, was also awarded his pin by incoming chairman, Robert J. Linney, as his first official act in office.

Principal speaker for the evening was Carter L. Burgess, who is president of American Machine and Foundry Co., New York. His provocative talk which closed the conference was entitled *Opportunity or Danger*.

Editor



COMMITTEE CHAIRMEN ARE BUSY MAKING MEETING PLANS

The important process of program planning within the Coal Division is again in motion as ideas and plans are rapidly forming for future meetings. *Coal News* readers will want to mark their calendars from the *Coming Events* column and make their reservations according to monthly announcements by committee chairmen.

Tentative plans are taking shape for the Bedford Springs meeting in September. The conference is being sponsored by the Coal and Industrial Minerals Divisions and the dates have been set for Sept. 24 to 26, 1959.

All major committee assignments are now filled and everyone is hard at work making arrangements before the June 15 deadline. As definite reservations and programs plans are established they will be announced by the men responsible for each phase of the conference.

The general conference co-chairmen are Edward J. Fox and John J. Schanz, Jr. Technical program committee co-chairmen are Leon W. Dupuy and H. E. Mauck.

Carlyle Gray is in charge of the field trip tentatively scheduled for Saturday morning. Golf and other recreational activities will be available for the afternoon. And the entertainment will be arranged by V. A. Stanton, while E. H. Johnson will handle the financial worries. T. S. Spicer and H. B. Charmbury are in charge of registration and arrangements. These men have been hand-picked to do each job well for a successful meeting. It's up to the members to follow through with early registration and whole-hearted support once the plans are made and the wheels set in motion.

In October the AIME-ASME Joint Solid Fuels Conference will take place in Cincinnati. Congratulations for a provocative program in 1958 were barely expressed before plans again were being formed for this year. The meeting will be held at the Netherland Hilton Hotel on Oct. 27 to 29, 1959. J. E. Tobey has been appointed chairman.

The annual meeting of the Rocky Mountain Coal Mining Inst. will be held at the Antlers Hotel, Colorado Springs, Colo., on June 28 to July 1.

Other meetings of interest to *Coal News* readers perhaps would include the Rock Mechanics Symposium sponsored by the Colorado School of Mines, the University of Minnesota, and Pennsylvania State University. Although it is not specifically concerned with coal, it covers general mining topics of interest to all who work underground. It is scheduled for April 20 to 22, 1959, and will include four sessions on the following general themes: 1) Factors common to comminution, underground failures, and failures resulting from explosions; 2) Factors common to soil mechanics and rock failure; 3) Seismology and explosions; and 4) Nuclear Blasts in Mining.

Finally, it is not too early to start thinking of the 1960 Annual Meeting in New York. H. Eugene Mauck is responsible for coordinating suggestions and plans for the sessions. He is program chairman for the Coal Div. for 1959-1960, in charge of the Annual Meeting arrangements. He would welcome any suggestions or ideas from the members on specific paper subjects, speakers, or session topics in general. If you have comments fresh in mind from this year's meeting, please send them along to Mr. Mauck at Olga Coal Co., Coalwood, W. Va. Individual participation on all levels is a vital part of successful program planning.

London Mining Meeting

The Institution of Mining Engineers is arranging a symposium on shaft sinking and tunneling to be held in London in July 1959. Papers will cover the general theme as it is demonstrated in many countries, including Australia, Canada, Germany, Great Britain, Holland, Poland, and the U. S. S. R. Papers will not be read in full, but will be circulated before the conference, in order to allow plenty of time for discussion of the subjects. Further information and application forms may be obtained by writing: The Institution of Mining Engineers, 3 Grosvenor Court, London S.W.1.

Managers Meet

The New Industrial Research Conference for 1959 has set itself a definite object, to find the solution of problems facing a manager of research. The workshop will be in session May 30 to June 5, 1959, meeting at Arden House in New York. Its first task will be to consider research management from the internal operation, the place of research, and research and service. The second task will be to consider the job of the research manager, the researcher, and problems of human relations.

A complete program and further details are available from Industrial Research Conference, 409 Engineering Building, Columbia University, New York 27, N. Y.

Advance registration only will be accepted for the period of the entire session. The fee is $525 which covers the cost of board and lodging and all books and materials used.

Laughing faces set the mood for the Utah Section's Dixieland Minstrel Show.

MINERS' REVIEW IS A MINSTREL SHOW

It would be hard to guess from these photographs who had more fun at the Annual Miners' Review, the 400 members and guests who packed the audience, or the black-faced minstrels themselves. But the outstanding success of the Utah Section's Dixieland Minstrel Show was plain to see, and resulted in everyone joining the fun. When talent and spirit and hard work are combined the rewards are great for all who take part.

AIME and WAAIME of the Utah Section joined forces on Feb. 7, 1959, for their third annual program. The Newhouse Hotel in Salt Lake City was the setting, and over 50 members took an active part on stage or backstage, with props, lights, sound, costumes and direction. Over 40 of this group had never taken part in any Annual Miners' Revue in the past. The Section tries each year to find new faces, new ideas, and fresh talent. The directors of the production were E. K. Olson, Jr., and Mrs. J. C. Landenberger, Jr.

Mrs. K. A. Lehner was in charge of choreography and helped originate some humorous steps. Mrs. T. J. Hubbard took care of the music, and Mrs. S. D. Michaelson managed the makeup, a mighty job for a minstrel show! Stage men were Harry C. Bauman and Roland Mulchay, and in charge of lights were J. D. Vincent, K. A. Lehner, and P. B. Petrie. To be commended for the colorful and authentic costumes is Mrs. Norman Weiss. And without the help of Norman Weiss, producer, and first vice chairman of the Section, the show would not have been possible.

The program began with the Interlocutor, Jim Van Stone, who introduced the End Men and Women: *Rastus*, played by Earshel W. Newman; *Cindy*, who was Mrs. Ed L. Anderson; *Amos*, alias Boris Ashurkoff; *Mose*, or Warren Woodward; *Mandy* played by Mrs. Wayne H. Burt; and *Sambo* who was really Sam Arentz.

Next came the lively Pickaninny Dancers, a highlight of the show. A variety of minstrels followed, interspersed with soft shoe dancing and the comical Half-Breed Whistlers who almost stole the show. The trio and duet presented some sharp satire of our political situation to add substance to the light-hearted program. And the chorus members gave spirited support to the acting cast. The hard-working singers and dancers in the chorus were: Claude Anderson; Fred Cunard; P. H. Ensign, Ed Moore; Burton Lyle; Ted Banta; J. C. Landenberger, Jr.; Clark Wilson; Bill Allison; Earl Allen; Mrs. Art Last; Mrs. Joe Ribotto; Mrs. L. W. Early; Mrs. O. E. House; Mrs. Warren Woodward; Mrs. Shy Evans; Shy Evans; Norman Weiss; and Hugo Johnson.

Singing minstrels pictured above are the Spiritual Trio of Bob Lawson, Sid Alderman, and Bob Lacy; and the duo of Mike Romney and Bob Bernick who sang the satire "Love Your Foreign Friends." In the picture below the chorus members give a spirited finale. The animated minstrels at the left, front row, are Sam Arentz, Boris Ashukoff, and E. M. Newman. In the back row are Warren Woodward, Mrs. Wayne A. Burt, Jim Van Stone, and Mrs. E. L. Anderson. They were the End Men and Women who opened the show, taking the roles of Sambo and Amos, Rastus, Mose, Mandy, the Interlocutor, and Cindy.

It is easy to see by these lively pictures the variety of fun offered by the Utah group. Starting at the left and moving clockwise, the before-and-after shots of the Half-Breed Whistlers attest to the ingenuity of the mining men. With hats on are Ewald Kipp and Ben Early. With hats off, above, they are joined by soloist Clark Wilson. A hearty welcome from Jim Van Stone opened the show. Snapped above in a typical Jolson pose, he is seen in the role of Interlocutor. Below him J. C. Landenberger is doing the old soft shoe called "Step'n Fetchit." Such talent unearthed by the mining crew made members wonder if they were in the wrong profession! Another surprise treat was the chorus line of Pickaninny Dancers caught below in their riotous routine. Behind those grinning masks are members of the Woman's Auxiliary: Mrs. David H. Curry, Jr.; Mrs. E. K. Olson, Jr.; Mrs. Jack W. Schultz; and Mrs. E. H. Snyder, Jr. And at the left, below, a trio bemoans a miner's woes in an original song called "Two Bit Copper and Ten Cent Zinc." The next line reads "How does a guy keep away from drink." Bill Allison and Shy Evans wield their tambourines, while Norman Weiss, kneeling, pleads for the good old days.

AIME NEWS

AIME Women Help Building Fund By Securing 7 Pledges

In the month of January the Member Gifts campaign for the United Engineering Center was averaging $56,000 per week in contributions. In five Founder Societies 56 local sections have exceeded their quotas, and a few have reached 200 pct. The highest percentage so far pledged is 88.3 pct for the AIChE. The Society with the largest quota to reach, AIEE, has the second highest percentage already pledged—72.7 pct.

AIME is fifth in the race to fulfill the quotas with 41.5 pct pledged. Our gifts included seven contributions totaling $3515 secured through the efforts of the Woman's Auxiliary. The AIEE Boston Section also has been helped by the women of their Auxiliary, and the campaign for funds has benefited from the support of the Society of Women Engineers.

The officers and leaders and all who gave in the Sections that exceeded their quotas are to be congratulated and sincerely thanked. AIME Sections that have gone over the top are: Gulf Coast (Houston), Hugoton, (Kansas), Montana, New York Petroleum, Ohio Valley, Oregon, Pennsylvania-Anthracite, Tri-State, Uranium, and Utah. The amount pledged by the Institute at the end of January totaled $207,810, and the total subscriptions in all the campaigns were $5,920,530, or 72.6 pct of the final goal of $8 million.

1958 Hoover Medal Awarded Wheeler

The Hoover Medal for 1958 was awarded to Lt. Gen. Raymond A. Wheeler at a dinner meeting of the Society of American Military Engineers on Feb. 19, 1959. Presentation was made at the Mayflower Hotel, Washington, D. C., by the president of the Society, Emerson C. Itschner. The Hoover Medal, outstanding engineering award of the United States, has been offered annually since 1930 by the four founder engineering societies: ASCE, AIME, ASME, and AIEE.

General Wheeler served in World War II as commander of the India-Burma Theater and Deputy Supreme Allied Commander of Southeast Asia Command; and as chief of the Army Engineers from 1945 until his retirement in 1949. He has also been associated with the International Bank for Reconstruction and Development, first as chief engineer and lately as consultant.

Montreal Greets UPADI in September

Montreal was the site of the fifth UPADI Convention Sept. 2 to 6, 1958, with the Engineering Inst. of Canada as host. K. F. Tupper was president. A conference on engineering education was the highlight of the meeting, with the aim of developing standards for engineering schools throughout the Hemisphere. Resolutions on internal organization, the exchange of information, and the establishment of regional centers for scientific studies totaled 47 in the business sessions. Headquarters will remain in Montevideo, and Argentina, Brazil, and the U. S. will remain on the board of direction. In 1960 the convention will be in Argentina, held in conjunction with a Pan Amercain Conference on Engineering Education.

Around the Sections

• The **Washington-Spokane** and **Idaho-Coeur d'Alene Sections** of the Woman's Auxiliary to AIME sponsored the women's program at the Northwest Mining Assn. convention in Spokane, on Dec. 5 and 6, 1958. Seven sessions and 32 speakers filled the regular program, with noted men including Royce Hardy, assistant secretary, Department of the Interior, and John Convey, Canadian director of mines, on the roster.

• The **University of Wyoming's** annual Engineer's Ball was held in Laramie on Dec. 6, 1958. One of the coeds representing the student engineering societies in a contest for queen of the ball was Katy Kugland, choice of the AIME delegates.

• New chairman of the **Utah Section** is Glen A. Burt, assistant to the manager of the International Smelting & Refining Co., Salt Lake City. First vice chairman is Norman Weiss, chief milling engineer, American Smelting & Refining Co. Oscar A. Glaeser, vice president and general manager, U. S. Smelting, Refining & Mining Co., was elected second vice chairman. The secretary-treasurer is Warren M. Woodward, Shell Oil Co.

The first meeting of the new year for the Utah Section was held at the Newhouse Hotel in Salt Lake City on January 15. Guest speaker was Harold Wright, executive vice president, The Galigher Co., who discussed rubbers and plastics.

• The **Mohawk-Hudson Section** met at Schenectady, N. Y., on December 12 in the Knolls research laboratory of General Electric Co. Bruce Chalmers, professor at Harvard University, spoke on *Structure of Large-Angle Grain Boundaries.*

• The **North Pacific Section** October meeting presented the following new officers: Ralph A. Watson, chairman; Merrill C. Teats, vice chairman; Donald L. Anderson, secretary-treasurer; and Marshall T. Huntting, correspondence secretary.

At the November meeting the featured speaker was John C. Kinnear, AIME Vice President.

• Nominations for 1959 officers took place at the November meeting of the **Mining Society of Southern California.** Henry Mulryan was nominated for chairman; H. P. Gower and Stanley Spellmeyer were chosen for vice chairmen; and C. W. Six was designated for treasurer, while Paul Patchek was named for the office of secretary. Councillor for a three-year term is Donald Carlisle.

• The **Bisbee-Douglas Subsection** of the Arizona Section was honored by an address given by J. D. Forrester, Dean of the College of Mines. The meeting was held November 13 at the Warren District Country Club.

• **Minnesota MBD Subsection** met at the Servicemen's Center in Chisholm, Minn., on December 3. W. J. Huston, chief of geology and ore research, Steep Rock Iron Mines Ltd., presented a talk on Steep Rock operations, including their concentrating problems, plant performance, history, and potential of the area.

• The **St. Louis Section** met at the Hotel York on January 9. Lawrence C. Widdoes, vice president, Inter-Nuclear Co., spoke on atomic energy applications for mining and for power at the first session.

At the February 13th meeting Marvin Breuer, U. S. Geological Survey, discussed *Photogrammetry and Topographic Mapping.*

• The January 14 meeting of the **El Paso Section** included a talk by William P. Kilgore, vice president of El Paso Industrial Council Inc., and elections for 1959. Mr. Kilgore spoke on *What's Ahead in Labor Relations.* New officers for the year are: Robert McGeorge, chairman; Eugene M. Thomas, vice chairman; Guy E. Ingersoll, secretary-treasurer; and directors: J. E. Douglas, Louis Laurel, B. D. Roberts, Earl Donahue, J. C. Rintelen, Jr., and A. A. Collins (Chihuahua).

On February 11th, the ballroom of the Hotel Cortez was the scene for a lecture on *The Ups and Downs of Mexico* by geology authority Tom Clendenin, American Smelting & Refining Co.

• The **San Francisco Section** held their January meeting on the 14th at the Engineers Club with dinner and a talk on uranium operations at Lucky Mc by John S. Anderson, manager, Utah Mining Corp. Mr. Anderson illustrated many phases of mining and ore treatment with colored slides.

• The annual breakfast of the **Colorado MBD Subsection** was held in Denver at the time of the National Western Mining Conference, Feb. 7, 1959. The scene was the Matchless Restaurant located in The Mile High Center.

• The first speaker of the new year for the **Washington, D. C., Section** was Louis McCabe, outgoing chairman. He spoke January 6 on *Air Pollution in the Mineral Industries.* The new officers elected in December are: Eugene D. Hardison, chairman; Richard W. Smith, John Croston, and J. T. Sherman, vice chairmen; Charles Merrill and W. F. Dietrich, council; Albert E. Schreck, secretary-treasurer, and his assistant, Alexander Garner.

On February 3rd the Section boasted "all the oysters you can eat" for their annual oyster party. The National Press Club was the setting for this feast.

• The **Oregon Section** started the new year with a trio of movies on the atom at the January 16 meeting. The new vice chairman and program chairman, S. L. Sampson, provided the films *Atomic Power, Atomic Energy,* and *The Atom Goes To Sea.*

• The **Reno Subsection** luncheon meeting on Jan. 9, 1959, featured speaker Hugh A. Shamberger, director, Nevada Dept. of Conservation, who spoke on Nevada water developments.

• The **Montana Section** met at the School of Mines on January 20 for a session on *Crystal Gazing With X-Rays.* James K. Grunig, research geologist, The Anaconda Co., illustrated the subject with slides, showing X-ray techniques in mining. The new officers for the Section are: J. J. Dougherty, chairman; George Hanson, vice chairman; Koehler Stout, secretary-treasurer; and executive committee members Guy Wever, Cliff Hicks, John Moore, and the student president of the Anderson-Carlisle Student Chapter at Montana School of Mines.

• The executive committee of the **Lehigh Valley Section** held its organizational meeting in Packer Hall, Lehigh University, on Jan. 23, 1959. N. Brown, chairman, announced the following appointments: B. H. Strom, publicity chairman; A. T. Kaufman, program chairman; G. A. Perhac, treasurer; and R. T. Renfrew, secretary.

• The **New York Section** heard Elmer W. Pehrson, Columbia University lecturer, discuss *Communist Minerals in Our Future.* Dinner at the Mining Club preceded the talk on January 22.

Personals

Ralph K. Gottshall, president of Atlas Powder Co., received an honorary doctor of science degree from Lafayette College, where he has been an active alumnus and vice president of the board of trustees.

C. E. Bartlett, formerly general superintendent of Compãnia Minera de Guatemala, has been doing consulting work the last three months and now is resident manager in British Guiana for Reynolds Metals Co.


C. E. BARTLETT


T. ROJAS

Tomas Rojas has been appointed assistant technical representative for Peru in the mining chemicals department of Cyanamid International.


C. F. SKINNER


J. COSGROVE

Charles F. Skinner has been named vice president and general manager of the Western-Knapp Engineering Co. Mr. Skinner has directed the firm's world wide engineering, design, and construction services as general manager since 1955.

John Cosgrove became district sales manager for Goodman Manufacturing Co. in the Huntington, W. Va. area, after several years in sales work with the company.


E. M. TITTMANN


K. H. DOERR, JR.

Edward McLanahan Tittmann has been elected vice president and director of American Smelting and Refining Co. In December Mr. Tittmann was made chairman of the board and chief executive officer of Southern Peru Copper Corp., in which Asarco holds a majority interest. **Kuno H. Doerr, Jr.,** was elected president of the Lima firm to replace Mr. Tittmann in that office. Mr. Doerr, Jr., had been president of the Garfield Chemical and Manufacturing Co. in Utah.

D. R. Dunham has been appointed manager of Columbia-Southern Chemical Corp. coal mine in Midvale, Ohio. Prior to this appointment he had been chief engineer for the firm in their New Jersey plant.

I. K. MacGregor is now vice president, staff operations, of American Metal Climax, Inc. Formerly he

PROPOSAL FOR AIME MEMBERSHIP

I consider the following person to be qualified for membership and request that a membership kit be mailed to him:

Name of Prospective Member: ....

Address ....

....

....

Name of AIME Member: ....

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SME

CHANGE OF ADDRESS AND PERSONALS FORM

CHANGING YOUR ADDRESS? Don't forget to notify us six (6) weeks before you move, if possible, to insure uninterrupted receipt of your publications and correspondence. Please fill in the form below and send it to: J. F. Lynch, Asst. Treasurer AIME, 29 West 39th Street, New York 18, N. Y.

Name ....

Old Address ....

....

....

New Address ....

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PERSONALS: Please list below your former company and title and your new title and company (or new work) for use in MINING ENGINEERING. (Copy deadline for personals items is six weeks before date of issue.)

Former Company ....

Former Title .... Length of Time There ....

New Company ....

New Title .... Date of Change ....

....

Any recent activity that would be of interest to members:

....

....

....

handled eastern operations of Climax Molybdenum Co. Div.

James W. Townsend is executive officer of minerals exploration field team in Region III, the Department of the Interior. Townsend, who has been employed in the Denver field office since 1951, will direct operations in an eight-state area.

International Minerals & Chemical Corp. has effected a consolidation which creates the following new positions: **George W. Moyers,** vice president of the new phosphate division; **I. M. LeBaron,** research vice president to direct the newly created staff unit. **Howard F. Roderick** has resigned to become vice president and director of Miles Laboratory, Elkhart, Ind. **William Bellano,** formerly in charge of the engineering division, was elected president of Gulf Sulphur Corp. The phosphate consolidation did not affect the following positions in the sales organization: **J. K. Westberg, L. W. Gopp,** and **S. T. Keel** remain sales managers.

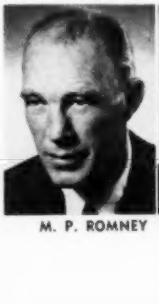
C. P. DONOHOE

R. B. McCORMICK

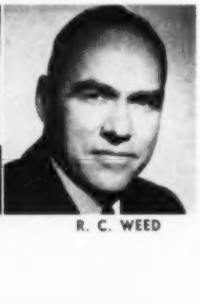
John F. Young, Frick Pa., has been appointed district mining engineer for U. S. Steel's Coal Div., Frick District.

Robert B. McCormick was appointed coordinator for chemical and ceramic materials in the Office of Minerals Mobilization, Dept. of the Interior. He has served with the munitions board and on the staff of the assistant secretary of defense for supply and logistics.

The Anaconda Co. announced personnel changes that included the retirement of **Albert Mendelsohn** as president and general manager of The Cananea Consolidated Copper Co., S. A., and the election of **Carroll P. Donohoe** to take his place. **Robert C. Weed,** formerly general superintendent, has been named vice president and assistant general manager. And **George E. Morris** has been elevated to the position of general superintendent.



M. P. ROMNEY



R. C. WEED

The Utah Mining Assn. announced new officers for 1959: **Lockwood W. Ferris,** president of Bonneville Ltd., was made president; **Oscar A. Glaeser,** vice president and general manager, western operations, U. S. Smelting Refining and Mining Co., became first vice president, and **S. K. Droubay,** vice president and general manager, United Park City Mines Co., became second vice president of the association. Officers who were re-elected are: **A. G. Mackenzie,** vice president and consultant; **Miles P. Romney,** secretary-manager; and **Walter M. Horne,** assistant secretary-manager.

Carl F. Austin, who was a student at the University of Utah, now has a research fellowship at New Mexico Bureau of Mines and Mineral Resources where he is engaged in explosives research.

E. G. OMAN

S. K. DROUBAY

Edward G. Oman has been appointed sales manager of the Western Ma-

personals

continued

chinery Co. office in Spokane after 15 years experience in mining and heavy equipment fields. **Jack H. How,** president of Western Machinery Co. and Western Knapp Engineering Co., has been elected president of the San Francisco Chamber of Commerce.

J. H. HOW — H. A. MYERS

H. A. Myers has been appointed operations manager of the Western Machinery Co. office in Denver. He had managed the Spokane office.

Election of nine Utah industrial leaders to the board of directors of Utah Manufacturers Assn. was announced in November. The new officers are as follows: **John Higginson,** Thiokol Chemical Corp.; **Royden G. Derrick,** president of Western Steel Co.; **Oscar A. Glaeser,** U. S. Smelting, Refining, Mining Co.; **L. K. Irvine,** manager of Utah Lumber Co.; **J. Arthur Wood,** president, Utah-Idaho Sugar Co.; **John B. Cahoon,** Interstate Brick Co.; **Ernest F. Goodner** American Gilsonite Co.; **F. Cooper Green,** Kennecott Copper Corp.; and **W. Rulon White,** president of the W. R. White Co.

Frank B. Jewett, Jr., has been elected executive vice president of Vitro Corp. of America and **William B. Hall** became vice president. Mr. Hall has been president of Vitro Uranium Co. in Salt Lake City. **Richard C. Cole** was promoted to president to succeed him there.

Richard C. Wells has been elected executive vice president of Freeport Sulphur Co. and continues as president of National Potash Co.

Francesco Audisio, formerly technical director of Societa Mineraria & Metallurgica di Pertusola, is now general manager of Penarroya-Maroc in Casablanca, Morocco.

Walter H. Ellis, Jr., became a geologist for Southwest Research Inst. He had been junior geologist for Anaconda Copper Co.

Richard Halstead, Jr., is now a mining engineer at Pacific Isle Mining Co., Hibbing, Minn. He was a student at South Dakota School of Mines and Technology, Rapid City, S. D.

Benjamin Lewon, sales engineer, has left Western Machinery Co. to join Smith Booth Usher Co. in Los Angeles.

W. J. Pollin, district manager for Utah Construction Co., is working on construction of the largest hydroelectric installation in Ecuador.

Fred Roehrer, sales engineer for Joy Manufacturing Co., has been transferred from the west Texas and New Mexico District to Arizona. His headquarters are in Phoenix.

Logan E. Davis, formerly senior application engineer, Joy Manufac-

Edgar C. Bain, retired vice president in charge of research and technology of the U. S. Steel Corp., was presented the Ambrose Monell Medal for distinguished achievement in mineral technology. The award was made at a dinner at Columbia University, by the Dean of Engineering, J. R. Dunning, right.

turing Co., now is manager of ventilation, dust collection and air conditioning division of Machinery Center, Inc. in Salt Lake City.

George Dirkes, mine and field geologist, has left Eagle-Picher for Condon-Cunningham Co. He has been studying Pleistocene and recent deposits in the midwest plus anhydrites of Oklahoma.

George H. Teal moved back to Boulder, Colo., after four years in the uranium fields of Utah Cord Co. He is now general manager of Tungsten Refining Co., Nonmetallics Div.

Robert W. Johnson has retired as president of Du Pont, S. A. de C. V. and Compãnia Mexicana de Explosives, S. A., after years of pioneering in explosives since he came to Mexico in 1925.

S. L. Rohrer was the guest speaker at the Ventura Chapter of the Western Mining Congress. He spoke on the trials and tribulations of a mining engineer. Mr. Rohrer writes that he has reached that stage of retirement where he can pick and choose jobs to his own specifications: "Chief specification being that I have no time for any property that isn't on or immediately adjacent to a GOOD fishing stream or lake!"

R. W. JOHNSON

S. L. ROHRER

Arnold B. Bower, Jr., has become a sales representative for the Metallurgical Products Dept. of General Electric Co. in the southern W. Va. area. He has assisted in development of new mining tools.

Carl A. Marshall, formerly managing director of The Fairmont Coal Bureau, has been concentrating on the eastern public utility market with a view to developing this new market for The Warner Collieries Co. which is opening a new mine in the Fairmont field. Mr. Marshall lives in Bridgewater, Conn.

F. A. McGonigle, vice president of Haile Mines Inc., became vice president of Howe Sound Co. in the merger of the two firms.

J. V. S. Norton, Bucyrus Erie Co., who was assistant export manager, now is assistant regional manager and has been visiting African open pit mines and English ironstone mines during the last year.

E. L. Oliver, Jr., formerly vice president of Dorr-Oliver Inc., has become vice president of Hodges Chemicals Co., Mt. View, Calif. Hodges has a new process to apply epoxy resins without a solvent.

J. L. BARRETT

F. MUNGER

James L. Barrett, formerly chief engineer and manager of manufacturing for Pennsylvania Drilling Co., has established the Geo-Drill Co., Bridgeville, Pa.

Fay Munger has been named manager, Mining Research and Development Div., The Jeffrey Manufacturing Co., Columbus, Ohio.

F. L. Beggs, production manager, has been elected a director of Carbola Chemical Co. Inc.

Robert M. Miller, mine foreman at Fronting Gold Mines, became program assistant for International

personals

continued

Cooperation Administration at the American Embassy in Mexico City.

F. Schultz graduated from Colorado School of Mines and is now an industrial engineer at Kennecott Copper Corp., Nevada Mines Div., in McGill, Nev.

C. M. GEORGE K. J. McDANIEL

Gardner-Denver Co. has announced three major personnel changes. **Charles M. George,** secretary, has been elected vice president and general manager for operations of the two plants at Quincy, Ill. **Aubrey H. Jones,** vice president and director of the export division, has been elected president of Gardner-Denver International, C. A. and **Kenneth J. McDaniel,** director of personnel, has become secretary.

E. M. Furness has been advanced to assistant executive vice president of Reserve Mining Co. He had been superintendent of Reserve's crushing and concentrating department. He has been in Silver Bay, Minn. since 1955.

A. H. JONES R. G. ALLEN

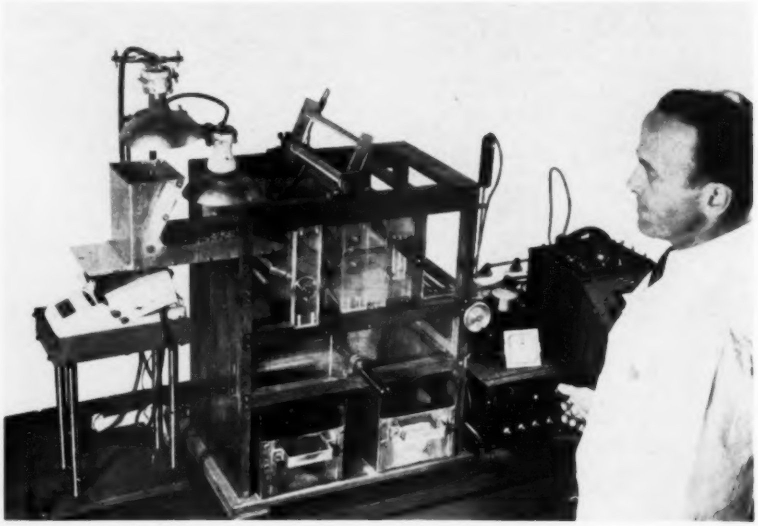
Bucyrus-Erie Co. elected **Robert G. Allen** president in charge of all operations to succeed **William L. Litle,** who continues as chairman of the board and senior officer. Other elections included **Victor C. Studley** as vice president in charge of finance; and **John R. Warner** as vice president in charge of purchasing. Mr. Studley, who also continues as treasurer, succeeds **Frederick C. Weiblen** who has retired.

Edward F. Ziolkowski, mining engineer, is now with Sunday Lake Iron Co. in a transfer of mines within the Pickands Mather organization. His home is in Ironwood, Mich.

Dennis D. Foley, formerly assistant division chief of Battelle Memorial Inst., has become the head of materials technology, nuclear power engineering, for Alco Products Inc. in Schenectady, N. Y.

C. R. Davis, Jr., has transferred to the Philadelphia district sales office of Chain Belt Co. He had been product supervisor, conveyor equipment, in Milwaukee, and now is district sales engineer.

Robert Y. Grant, who had been assistant director for industry in the ICA Mutual Security Mission to China, now is taking part in the U. S. Operations Mission to Djakarta.



A. A. Linari-Linholm is photographed with the laboratory model electrostatic diamond separator representing the separators operating at the Consolidated Diamond Mines of South West Africa Ltd. where he is the chief metallurgist.

Obituaries

Henry Krumb

An Appreciation By
H. DeWitt Smith

Henry Krumb, consulting mining engineer, died of cancer in New York City on Dec. 27, 1958, at the age of 83. As a youth he decided to become a consulting mining engineer and every step he took was with this goal definitely in view. His story is an inspiration to the engineering youth of this country, in whom he was vitally interested and for whom he set up the Henry Krumb Mining and Metallurgical Scholarship Fund at Columbia University.

Henry Krumb was born in Brooklyn, Nov. 15, 1875. He attended the Columbia University School of Mines, from which he graduated with the degree of Engineer of Mines in 1898.

After a year in New York City, first as draftsman and engineer and then as assayer and chemist, he went to British Columbia to look for a job more directly connected with mining, his chosen profession. He ran several small mines in the Rossland district, and in 1902 became chief engineer of John Hays Hammond.

In 1904 Mr. Krumb joined the engineering staff of the Guggenheim Exploration Co. as field engineer under Mr. Hammond, and during the next three years he examined and reported on gold, silver, copper, lead, and zinc mining properties in Alaska, Canada, Mexico, and various districts in the U. S. During this period Mr. Krumb began his connection with the so-called porphyry copper mines, the most important work of his career. He is universally credited among engineers with playing an important part in the successful development of the low-grade copper deposits which today yield about 60 pct of the world's copper.

In 1905 he examined the Utah Copper mine at Bingham, Utah, for the Guggenheim group. This examination required eight months' work, including extensive underground development and diamond drilling to confirm the extent of the orebody, and large-scale treatment tests. As the result of Mr. Krumb's report, the Guggenheim furnished the funds with which shovel operations were started at the mine and a large concentrating plant erected at Garfield. This was the first porphyry mine to be exploited on a large scale and is today the largest copper mine in North America.

Immediately after the Utah Copper financing, Mr. Krumb examined

other similar properties, including the Nevada Consolidated, and Cumberland Ely, in Nevada. A controlling interest in the Nevada properties was acquired by the Guggenheims as a result of Mr. Krumb's recommendations, a concentrator and smelter built at McGill, and a railroad built 140 miles to the main line of the Southern Pacific Railroad.

In the summer of 1906, Mr. Krumb headed an expedition to investigate the copper resources of the Copper River district in Alaska. As a result of these examinations, Kennecott Mines Co. was financed by the Guggenheim and J. P. Morgan & Co. to develop the mines and to build the Copper River and Northwestern Railroad.

In 1907, at the age of 32, Mr. Krumb established himself in private practice as consulting mining engineer, with offices in Salt Lake City, Utah. The next year he introduced the use of churn drills in the development of porphyry copper mines. Using this method, in the next two years he developed ore reserves of Ray Consolidated and Gila River Cos. in Arizona from 1 million to 70 million tons. At about the same time Mr. Krumb negotiated the purchase for William Boyce Thompson of the prospective porphyry properties of the Inspiration Copper Co., near Globe, Ariz. Mr. Thompson provided $11 million for the development and equipment of these properties and Mr. Krumb served as consulting engineer until 1912, during which time 45 million tons of ore were developed.

Mr. Krumb in 1909 examined a prospect known as the Silver Queen, near Superior, Ariz., and on his favorable report, Mr. Thompson organized the Magma Copper Co. to acquire and develop it. Mr. Krumb served as consulting engineer for several years for this company, and later as a director.

Mr. Krumb's examination of Kennecott's Alaskan property in 1915 disclosed an orebody averaging 70 pct copper, the most phenomenal deposit of rich copper ore ever discovered. Mr. Krumb recommended that Kennecott use much of its great earnings from this high grade but limited orebody in acquiring large low grade ore reserves of the porphyry type, to assure long life to the company. Following his recommendation, Kennecott Copper Corp. was formed, the Alaskan mines and the Copper River Railroad absorbed, the Braden Copper Co. in Chile merged and financed, and a large block of Kennecott shares exchanged for Utah Copper shares. Later, additional Utah Copper shares were obtained in exchange for Kennecott shares and by open market purchases. Ultimately all of Utah Copper and the Nevada Consolidated, Ray and Chino companies were absorbed into the great Kennecott Copper Corp., as the result of discussions initiated by Mr. Krumb with Mr. Stephen Birch, president of Kennecott, on a hillside below Kennecott's Jumbo mine one summer afternoon in 1915.

In 1916, Mr. Krumb went to South America to examine the Braden

Necrology

Date Elected	Name	Date of Death
1944	Lawrence W. Allen	July 1958
1936	Leland C. Ball	June 23, 1958
1914	J. T. Boyd	Jan. 3, 1959
1946	Cecil H. Desch Honorary Member	June 19, 1958
1958	Albert R. Eckel	Dec. 4, 1958
1919	Gordon Dugald	Oct. 3, 1958
1885	H. L. Hollis Legion of Honor	Nov. 20, 1958
1928	H. O. Howard	Unknown
1915	L. R. Jenkins	Dec. 25, 1957
1913	Lee O. Kellogg	Dec. 2, 1958
1911	Joseph E. Kennedy	Jan. 9, 1959
1937	J. A. Kingsbury	Nov. 19, 1958
1905	Henry Krumb Honorary Member Legion of Honor	Dec. 27, 1958
1917	J. F. Linthicum	Nov. 17, 1958
1939	R. C. McQuire	Dec. 23, 1958
1919	Wendell Z. Miller	Mar. 11, 1958
1920	R. M. Overbeck	Sept. 23, 1958
1916	C. O. Stee	Dec. 28, 1958
1943	Charles E. Tonry	Dec. 21, 1958
1915	B. H. Van Der Linden	Oct. 5, 1958
1915	H. T. Walsh	Oct. 10, 1958
1895	William Y. Westervelt Legion of Honor	Oct. 8, 1958
1926	B. L. Wheeler	Jan. 4, 1958
1935	Howard G. Wilcox	Sept. 9, 1958

property, the Chuquicamata mine of Chile Copper Co., and the mines of Cerro de Pasco in Peru. During World War I, he was a member of the Priorities Committee of the War Industries Board in Washington.

In 1920, Mr. Krumb examined the Flin Flon property in northern Manitoba, for a group, of which William Boyce Thompson, who later organized Newmont Mining Corp., was a member. The option was allowed to expire in the depression of 1921, but in 1928 Mr. Krumb was instrumental in obtaining a substantial holding in the property for certain of the clients for whom he made the original examination in 1920. Hudson Bay Mining & Smelting Co. Ltd., was organized and $24 million spent on development and equipment of this property. Mr. Krumb had been a director and vice president of this company since its incorporation.

He was an active director of Newmont Mining Corp. from 1925 until his death, of O'okiep Copper Co. Ltd. from 1937, and of Newmont Oil Co. from 1939.

Henry Krumb became a member of the AIME in 1905, and served as vice president from 1928 until 1942. His services to the Institute and to his profession through this period are summarized in the citation given him in 1939 when he was elected Honorary Member of the Institute "in recognition of his notable attainments as an engineer; of his constant and effective efforts in behalf of the profession, and his neverfailing and wise counsel in the administration of Institute affairs during critical years."

In 1938 a donor, whose identity was disclosed only to the Treasurer of the AIME, established a special fund designated Endowment Fund "X", with total gifts to principal of $96,912 over a period of ten years. The donor expressed the wish that the income from this fund be used preferably only in times of economic depression, so that the Institute would not be forced to curtail its activities when income from other sources dwindled. One objective of the donor was to show the soundness of investing in common stocks of well-operated mining and oil companies. This is borne out by the fact that the market value of this fund as of Dec. 31, 1958, was $860,272. After his death it was disclosed that Mr. Krumb was the anonymous donor of Fund "X", henceforth to be known as the Henry Krumb Endowment Fund.

In 1939 he was awarded the Egleston Medal of Columbia University for "Distinguished Engineering Achievement." In 1951 he received the Honorary Degree of Doctor of Science from Columbia University, and in 1956 he was awarded the Columbia Class of 1889 Medal for "Eminent Achievement." He served on the board of trustees of Columbia University from 1941 to 1947.

Mr. Krumb had four major interests in his professional life—his career as a mining engineer, the advancement of the younger members of his profession, the AIME, and his alma mater, Columbia University. He had remarkable technical and financial acumen, a pleasing but forceful way in extracting basic facts and a most retentive memory. His fund of knowledge and his definite views on business, finance, and politics made him an equally welcome companion on the trail or across a directors' table. He was a great and very modest man. He mapped out his course of life and followed it to the end.

Mr. Krumb is survived by his wife, Mrs. LaVon Duddleson Krumb, to whom he was married in San Francisco in 1914.

Membership

Proposed for Membership
Society of Mining Engineers of AIME

Total AIME membership on Jan. 31, 1959, was 30,710; in addition 3,819 Student Members were enrolled.

ADMISSIONS COMMITTEE

A. E. Weissenborn, Chairman; Frank A. Ayer, Vice Chairman; R. T. Lassiter, Charles H. Lambur, Alfred D. Rood, William J. Rude, L. P. Warriner.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

Members

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John R. Grizzle, Benicia, Calif.
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Russell J. Hayden, Boulder City, Nev.
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Sidney C. Howell, Duluth, Minn.
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Robert L. L'Esperance, Sewickley, Pa.
O. P. Handa, Dhanbad, India
Delbert J. Merrill, Rexford, N. Y.
Donald H. Meyers, Hibbing, Minn.
Raymond C. Nispel, Ruth, Nev.
Moss Patterson, Madisonville, Ky.
Robert P. Pearsall, Jr., Virginia, Minn.
James F. Poulos, Lakeview, Ore.
Brune Scipioni, Virginia, Minn.
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Eugene D. Smith, Glendale, Calif.
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Garth W. Thornburg, Lakeview, Ore.
Edward P. Utting, Cottesloe, W. Australia
Anthony T. Vellella, Hibbing, Minn.
Keith G. Wallace, Los Altos, Calif.
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CHANGE OF STATUS

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John R. Harmon, Yerington, Nev.
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Ben H. Slothower, Salt Lake City
James R. Stringham, Santa Barbara, Mexico
Conrad A. Wojy, Dover, N. J.
William G. Wood, La Oroya, Peru

Junior to Member

Raymond G. Garcia, Lakeland, Fla.
John W. Goth, Chicago

REINSTATEMENTS

Member

George K. Bjorklund, Haywood, Calif.
William L. Emerick, Tucson, Ariz.

REINSTATEMENT—CHANGE OF STATUS

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John P. Manning, Jr., South Salem, N. Y.
James B. Morrow, Hibbing, Minn.

Junior to Member

Blucher F. Allison, Fayette Co., W. Va.
Sam Christo, Jr., Carlsbad, N. M.
Chester D. Vaughan, Jr., Lansdale, Pa.

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Space limited to AIME members or to companies that have at least one member on their staffs. One inch, $50 per year; half inch, $30 per year, payable in advance.

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